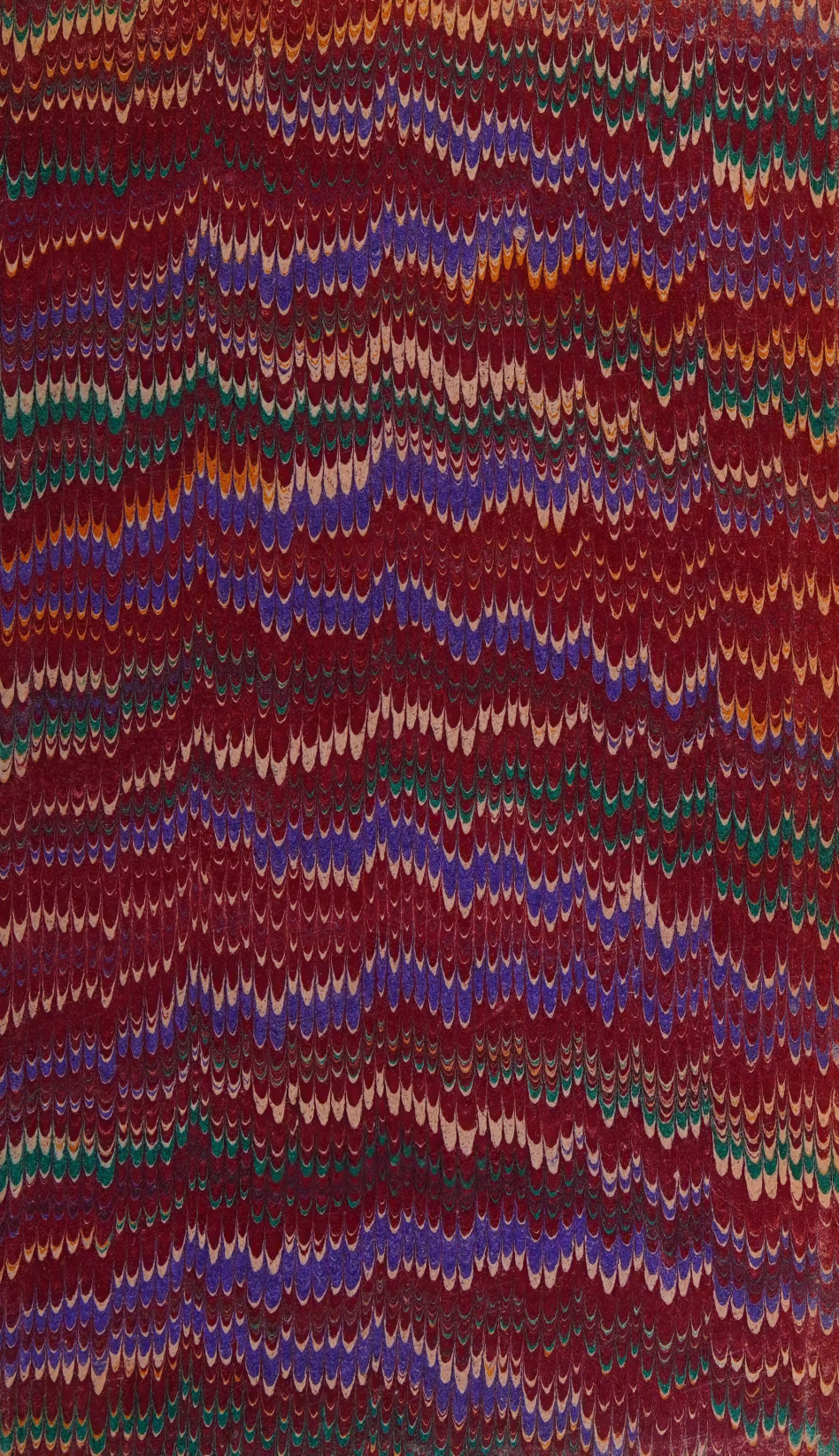

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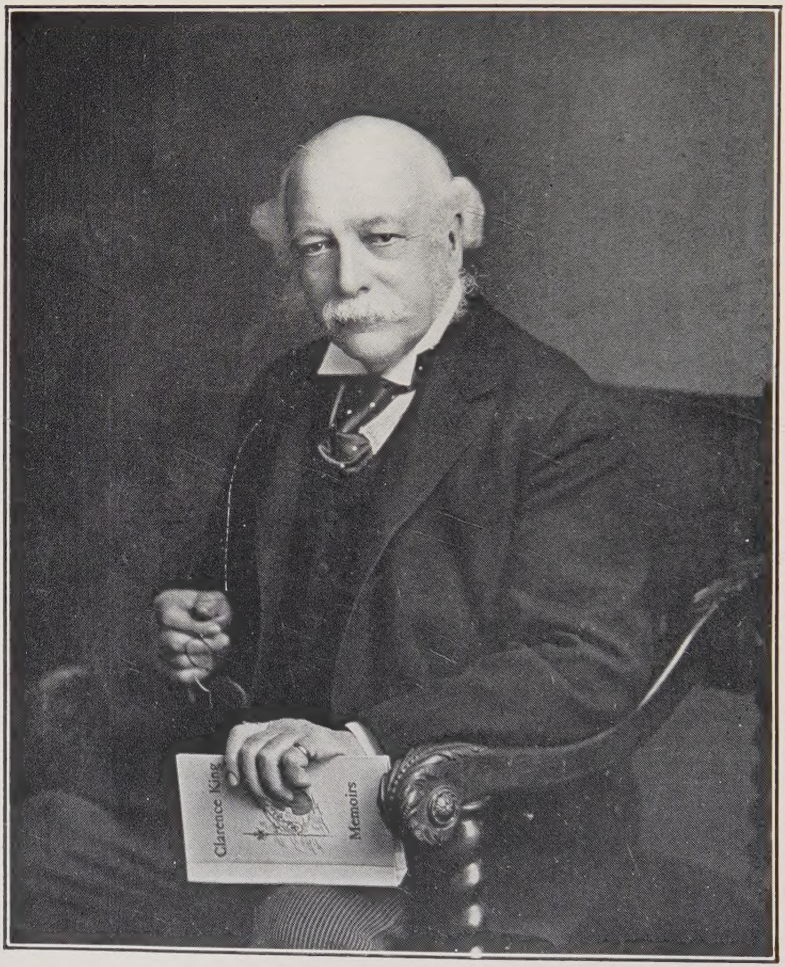

James D. Hague

See Biographical Notice, p. 677.

TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING ENGINEERS.

VOL. XXXIX.

CONTAINING THE PAPERS AND DISCUSSIONS OF 1908.

NEW YORK, N. Y.:
PUBLISHED BY THE INSTITUTE,
AT THE OFFICE OF THE SECRETARY.
1909.

PREFACE.

This volume comprises all the proceedings, papers, and discussions of the Institute published during 1908, with the following exceptions:

1. Brief obituary notices of members and associates reported as deceased during 1907[1] and 1908.[2]

2. Library reports, including a list of the serial publications in the libraries of the American Institute of Electrical Engineers, the American Society of Mechanical Engineers, and the American Institute of Mining Engineers,[3] and other announcements of general but temporary interest, furnished to members in *Bi-Monthly Bulletin*, Nos. 19 to 24, during the year 1908.

3. Account of the excursions and entertainments connected with the New York meeting, February, 1908,[4] and with the Chattanooga meeting, October, 1908,[5] including in the latter the visit to Ducktown and the mines and works of the Tennessee Copper Co.

4. Special pamphlets for use during visit to the tunnel-work in New York and vicinity at the time of the New York meeting in February, 1908—namely:

The Tunnels of the Hudson Companies, by S. D. V. Burr, New York, N. Y.[6]

The Manhattan Cross-Town Tunnels of the Pennsylvania Railroad.[7]

New York Tunnel Extension of the Pennsylvania Railroad System, by William Couper, New York, N. Y.[8]

5. Report of Committee 24 of the International Association for Testing Materials, entitled The Uniform Nomenclature of Iron and Steel.[9]

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[1] *Bi-Monthly Bulletin*, No. 19, January, 1908, pp. xlix to lxxv.

[2] *Ibid.*, No. 20, March, 1908, pp. lvii to lxvii; No. 21, May, 1908, pp. xlvii to li; No. 22, July, 1908, pp. xxxvii to xliii; No. 23, September, 1908, pp. xxix to xxxiii; No. 24, November, 1908, pp. xxxiii to xxxvi, and *Bulletin* No. 25, January, 1909, pp. xviii to xxvii.

[3] *Ibid.*, No. 19, January, 1908, pp. xxiii to xlii.

[4] *Ibid.*, No. 20, March, 1908, pp. 243 to 247.

[5] *Ibid.*, No. 24, November, 1908, pp. 1191 to 1210.

[6] *Ibid.*, No. 20, March, 1908, pp. 253 to 274.

[7] *Ibid.*, No. 20, March, 1908, pp. 275 to 288.

[8] *Ibid.*, No. 22, July, 1908, pp. xlv to lii.

[9] *Ibid.*, No. 20, March, 1908, pp. 227 to 237.

6. A paper entitled, The Investigations of Fuels and Structural Materials by the Technologic Branch of the U. S. Geological Survey, by Joseph A. Holmes, Washington, D. C.[10]

7. A paper entitled, The Clinton Iron-Ore Deposits in Alabama, by Ernest F. Burchard, Washington, D. C.[11] (To be published in *Transactions*, Vol. XL.)

8. A paper entitled, The Clinton Iron-Ore Deposits in Stone Valley, Huntingdon County, Pa., by J. J. Rutledge.[12] (Held for Vol. XL.)

9. A paper entitled, The Clinton Iron-Ore Deposits in New York State, by D. H. Newland, Albany, N. Y.[13] (Held for Vol. XL.)

10. A paper entitled, Studies of Illinois Coals, by H. Foster Bain and others.[14] (Held for Vol. XL.)

11. A paper entitled, Ozark Lead- and Zinc-Deposits: Their Genesis, Localization, and Migration, by Charles R. Keyes, Des Moines, Iowa.[15] (Held for Vol. XL.)

12. Year Book, containing a List of Members and Associates, revised to Jan. 1, 1909, pp. 161. Published in separate form and distributed with *Bulletin* No. 25, January, 1909.

The publications, thus mentioned above, exceed 500 pages of valuable material, which could not be included in this volume of the *Transactions* on account of lack of space.

On the other hand, this volume includes:

1. A paper entitled, South African Tin-Deposits, by William R. Rumbold, Oruro, Bolivia, S. A., presented at the New York meeting, April, 1907, but revised proof of which was not received from the author in time to incorporate in Vol. XXXVIII.

2. One discussion of a paper, which was received after Jan. 1, 1909, yet in time to be here printed and not held over for Vol. XL.

JOSEPH STRUTHERS,
Assistant Secretary and Editor.

[10] *Bi-Monthly Bulletin*, No. 22, July, 1908, pp. 531 to 550; also in more expanded form in a *Bulletin* of the U. S. Geological Survey (now in preparation).

[11] *Ibid.*, No. 24, November, 1908, pp. 997 to 1055.

[12] *Ibid.*, No. 24, November, 1908, pp. 1057 to 1087.

[13] *Bulletin* No. 27, March, 1909, pp. 265 to 283.

[14] *Bi-Monthly Bulletin*, No. 24, November, 1908, pp. 1099 to 1170.

[15] *Bulletin* No. 26, February, 1909, pp. 119 to 166.

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OFFICERS.

For the year ending February, 1909.

COUNCIL.*

PRESIDENT OF THE COUNCIL.

JOHN HAYS HAMMOND.....NEW YORK, N. Y.
(Term expires February, 1909.)

VICE-PRESIDENTS OF THE COUNCIL.

SAMUEL B. CHRISTY.....BERKELEY, CAL.
JOHN A. CHURCH.....NEW YORK, N. Y.
PERSIFOR FRAZER.....PHILADELPHIA, PA.
(Term expires February, 1909.)

J. PARKE CHANNING.....NEW YORK, N. Y.
FREDERICK W. DENTON.....PAINESDALE, MICH.
JOHN B. FARISH.....NEW YORK, N. Y.
(Term expires February, 1910.)

COUNCILORS.

THEODORE DWIGHT.....NEW YORK, N. Y.
WALTER WOOD.....PHILADELPHIA, PA.
WM. FLEET ROBERTSON.....VICTORIA, B. C., CAN.
(Term expires February, 1909.)

B. F. FACKENTHAL, JR.....EASTON, PA.
H. O. HOFMAN.....BOSTON, MASS.
WALTER R. INGALLS.....NEW YORK, N. Y.
(Term expires February, 1910.)

C. R. CORNING.....NEW YORK, N. Y.
R. V. NORRIS.....WILKES-BARRE, PA.
WILLIAM H. SHOCKLEY.....TONOPAH, NEV.
(Term expires February, 1911.)

SECRETARY OF THE COUNCIL.

R. W. RAYMOND.....29 W. 39th St., NEW YORK, N. Y.
(Term expires February, 1909.)

ASSISTANT SECRETARY AND EDITOR.

JOSEPH STRUTHERS.....NEW YORK, N. Y.

CORPORATION.

JAMES GAYLEY, President; JAMES DOUGLAS, Vice-President;
R. W. RAYMOND, Secretary; FRANK LYMAN, Treasurer;
JOSEPH STRUTHERS, Assistant Secretary and Assistant Treasurer.

DIRECTORS.

THEODORE DWIGHT, CHARLES H. SNOW, R. W. RAYMOND.
(Term expires February, 1909.)

JAMES GAYLEY, CHARLES KIRCHHOFF, FRANK LYMAN.
(Term expires February, 1910.)

JAMES DOUGLAS, JAMES F. KEMP, ALBERT R. LEDOUX.
(Term expires February, 1911.)

Consulting Attorneys, Blair & Rudd, New York, N. Y.

* SECRETARY'S NOTE.—The Council is the professional body, having charge of the election of members, the holding of meetings (except business meetings), and the publication of papers, proceedings, etc. The Board of Directors is the body legally responsible for the business management of the Corporation, and is therefore, for convenience, composed of members residing in New York.

OFFICERS ELECTED AT ANNUAL MEETING, FEB. 16, 1909.

The list of officers on the opposite page is that of the year 1908, which is the period covered by the contents of this volume of the *Transactions*. But the result of the election at the Annual Business Meeting, February, 1909, although strictly belonging to the next volume, is here published for the convenience of members.

The following officers were elected by vote of the members and associates in person or by proxy at the Annual Meeting, Feb. 16, 1909:

COUNCIL.

President of the Council.

D. W. Brunton, ... Denver, Colo.

(To serve for one year. Term expires February, 1910.)

Vice-Presidents of the Council.

W. C. Ralston, ... San Francisco, Cal.
W. L. Saunders, ... New York, N. Y.
H. V. Winchell, ... St. Paul, Minn.

(To serve for two years. Term expires February, 1911.)

Councilors.

Karl Eilers, ... New York, N. Y.
Alex. C. Humphreys, ... New York, N. Y.
Wm. G. Miller, ... Toronto, Can.

(To serve for three years. Term expires February, 1912.)

Secretary of the Council.

R. W. Raymond, ... New York, N. Y.

(To serve for one year. Term expires February, 1910.)

Assistant Secretary and Editor (by appointment).

Joseph Struthers, ... New York, N. Y.

(To serve for one year. Term expires February, 1910.)

DIRECTORS OF THE CORPORATION.

Theodore Dwight, R. W. Raymond, and Charles H. Snow.

(To serve for three years. Term expires February, 1912.)

The following are the officers of the corporation for the year ending February, 1910:

President, James Gayley, New York, N. Y.
Vice-President, James Douglas, New York, N. Y.
Secretary, R. W. Raymond, New York, N. Y.
Treasurer, Frank Lyman, New York, N. Y.
Assistant Secretary and Assistant Treasurer, Joseph Struthers, New York, N. Y.

PAST OFFICERS.

Presidents.

*David Thomas	1871
R. W. Raymond	1872–1874
*A. L. Holley	1875
*Abram S. Hewitt	1876
*T. Sterry Hunt	1877
*Eckley B. Coxe	1878–1879
*William P. Shinn	1880
William Metcalf	1881
*Richard P. Rothwell	1882
Robert W. Hunt	1883
James C. Bayles	1884–1885
Robert H. Richards	1886
*Thomas Egleston	1887
William B. Potter	1888
Richard Pearce	1889
*Abram S. Hewitt	1890
John Birkinbine	1891–1892
H. M. Howe	1893
John Fritz	1894
*J. D. Weeks	1895
E. G. Spilsbury	1896
*Thomas M. Drown	1897
Charles Kirchhoff	1893
James Douglas	1899–1900
E. E. Olcott	1901–1902
Albert R. Ledoux	1903–1904
James Gayley (Council)	1905
James Gayley (Corporation)	1905 ——
Robert W. Hunt (Council)	1906
John Hays Hammond (Council)	1907–1908

Secretaries.

*Martin Coryell	1871–1872
*Thomas M. Drown	1873–1884
R. W. Raymond	1884 ——

Treasurers.

J. Pryor Williamson	1871–1872
*Theodore D. Rand	1872–1903
Frank Lyman	1903 ——

* Deceased.

HONORARY MEMBERS.

Prof. Richard Åkerman	Stockholm, Sweden.
Andrew Carnegie	New York, N. Y.
Dr. James Douglas	New York, N. Y.
Prof. Haton de la Goupillière	Paris, France.
Sir Robert A. Hadfield	London, England.
Prof. Hans Hoefer	Leoben, Austria.
Prof. Henri Louis Le Chatelier	Paris, France.
M. Floris Osmond	Saint Leu, France.
John E. Stead	Middlesbrough, England.
Prof. Dimitry Constantin Tschernoff	St. Petersburg, Russia.
Charles D. Walcott	Washington, D. C.

Honorary Members (*Deceased*).

	Year of Decease.
Bell, Sir Lowthian	1904
Castillo, A. Del	1895
Contreras, Manuel Maria	1902
Daubrée, A	1896
Drown, Thomas M	1904
Gaetzschmann, Moritz	1895
Gruner, L	1883
Hunt, T. Sterry	1892
Kerl, Bruno	1905
Le Conte, Joseph	1901
Lesley, J. P	1896
Patera, Adolph	1890
Percy, John	1889
Posepny, Franz	1895
Richter, Theodor	1898
Roberts-Austen, W. C	1902
Serlo, Albert	1898
Siemens, C. William	1883
Thomas, David	1882
Tunner, Peter R. von	1897
Wedding, Hermann	1908

LIST OF THE MEETINGS OF THE INSTITUTE AND THEIR LOCALITIES FROM ITS ORGANIZATION TO OCTOBER, 1908.

No.	Place.	Date.	Vol.	Trans. Page.
1.	Wilkes-Barre, Pa.*	May, '71	1	3
2.	Bethlehem, Pa	Aug., '71	1	10
3.	Troy, N. Y.	Nov., '71	1	13
4.	Philadelphia, Pa	Feb., '72	1	17
5.	New York, N. Y.*	May, '72	1	20
6.	Pittsburg, Pa	Oct., '72	1	25
7.	Boston, Mass	Feb., '73	1	28
8.	Philadelphia, Pa.*	May, '73	2	3
9.	Easton, Pa	Oct., '73	2	7
10.	New York, N. Y	Feb., '74	2	11
11.	St. Louis, Mo.*	May, '74	3	3
12.	Hazleton, Pa	Oct., '74	3	8
13.	New Haven, Conn	Feb., '75	3	15
14.	Dover, N. J.*	May, '75	4	3
15.	Cleveland, O	Oct., '75	4	9
16.	Washington, D. C	Feb., '76	4	18
17.	Philadelphia, Pa.†	June, '76	5	3
18.	Philadelphia, Pa	Oct., '76	5	19
19.	New York, N. Y	Feb., '77	5	27
20.	Wilkes-Barre, Pa.*	May, '77	6	3
21.	Amenia, N. Y	Oct., '77	6	10
22.	Philadelphia, Pa	Feb., '78	6	18
23.	Chattanooga, Tenn.*	May, '78	7	3
24.	Lake George, N. Y	Oct., '78	7	103
25.	Baltimore, Md.*	Feb., '79	7	217
26.	Pittsburg, Pa	May, '79	8	3
27.	Montreal, Canada	Sept., '79	8	121
28.	New York, N. Y.*	Feb., '80	8	275
29.	Lake Superior, Mich	Aug., '80	9	1
30.	Philadelphia, Pa.*	Feb., '81	9	275
31.	Staunton, Va	May, '81	10	1
32.	Harrisburg, Pa	Oct., '81	10	119
33.	Washington, D. C.*	Feb., '82	10	225
34.	Denver, Colo	Aug., '82	11	1
35.	Boston, Mass.*	Feb., '83	11	217
36.	Roanoke, Va	June, '83	12	3
37.	Troy, N. Y	Oct., '83	12	175
38.	Cincinnati, O.*	Feb., '84	12	447
39.	Chicago, Ill	May, '84	13	1
40.	Philadelphia, Pa	Sept., '84	13	285
41.	New York, N. Y.*	Feb., '85	13	585
42.	Chattanooga, Tenn	May, '85	14	1
43.	Halifax, N. S	Sept., '85	14	307
44.	Pittsburg, Pa.*	Feb., '86	14	587
45.	Bethlehem, Pa	May, '86	15	lxiii.
46.	St. Louis, Mo	Oct., '86	15	lxx.
47.	Scranton, Pa.*	Feb., '87	15	lxxvii.
48.	Utah and Montana	July, '87	16	xvii.
49.	Duluth, Minn	July, '87	16	xxiv.
50.	Boston, Mass.*	Feb., '88	16	xxviii.
51.	Birmingham, Ala	May, '88	17	xix.
52.	Buffalo, N. Y	Oct., '88	17	xxiv.
53.	New York, N. Y.*	Feb., '89	17	xxxi.
54.	Colorado	June, '89	18	xvii.
55.	Ottawa, Canada	Oct., '89	18	xxiv.
56.	Washington, D. C.*	Feb., '90	18	xxx.
57.	New York, N. Y	Sept., '90	19	vii.
58.	New York, N. Y.*	Feb., '91	19	xxv.
59.	Cleveland, O	June, '91	20	xvi.
60.	Glen Summit, Pa	Oct., '91	20	lxi.
61.	Baltimore, Md.*	Feb., '92	21	xix.
62.	Plattsburgh, N. Y	June, '92	21	xxxiii.
63.	Reading, Pa	Oct., '92	21	xliv.
64.	Montreal, Canada*	Feb., '93	21	lii.
65.	Chicago, Ill	Aug., '93	22	xiii.
66.	Virginia Beach, Va.*	Feb., '94	24	xvii.
67.	Bridgeport, Conn	Oct., '94	24	xxxv.
68.	Florida‡	Mar., '95	25	xix.
69.	Atlanta, Ga	Oct., '95	25	xxxiii.
70.	Pittsburg, Pa.*	Feb., '96	26	xvii.
71.	Colorado	Sept., '96	26	xxix.
72.	Chicago, Ill	Feb., '97	27	xvii.
73.	Lake Superior	July, '97	27	xxx.
74.	Atlantic City, N. J.*	Feb., '98	28	xvii.
75.	Buffalo, N. Y	Oct., '98	28	xxxvi.
76.	New York, N. Y.*	Feb., '99	29	xvii.
77.	California	Sept., '99	29	xlix.
78.	Washington, D. C.*	Feb., '00	30	xix.
79.	Canada	Aug., '00	30	xlv.
80.	Richmond, Va.*	Feb., '01	31	xix.
81.	Mexico	Nov., '01	32	cxviii.
82.	Philadelphia, Pa.§	May, '02	33	xxxv.
83.	New Haven, Conn	Oct., '02	33	xlvii.
84.	Albany, N. Y.*	Feb., '03	34	xxiii.
85.	New York, N. Y	Oct., '03	34	lxi.
86.	Atlantic City, N. J.*	Feb., '04	35	xxiii.
87.	Lake Superior	Sept., '04	35	xlii.
88.	Washington, D. C	May, '05	36	xlii.
89.	British Columbia	July, '05	36	liii.
90.	Bethlehem, Pa	Feb., '06	37	xli.
91.	London, England	July, '06	37	xlviii.
92.	New York, N. Y	April, '07	38	lii.
93.	Toronto, Canada	July, '07	38	lix.
94.	New York, N. Y	Feb., '08	39	xli.
95.	Chattanooga, Tenn	Oct., '08	39	xlviii.

* Annual meeting for the election of officers. The rules were amended at the Chattanooga meeting, May, 1878, changing the annual election from May to February.

† Begun in May at Easton, Pa., for the election of officers, and adjourned to Philadelphia.

‡ Begun in February at New York City, for the election of officers, and adjourned to Florida.

§ " " " " " " " " " " " " " " to Philadelphia.

PUBLICATIONS.

THE publications of the Institute comprise:

TRANSACTIONS.

The volumes of *Transactions*, which are published annually, contain the list of officers, rules, etc., the Proceedings, and the papers revised for final publication. (In this revision, after the preliminary publication, authors are permitted to use the largest liberty; and the changes and additions made in papers are sometimes important. It should be borne in mind by those who study or quote a paper in the preliminary edition, that they may not have in that form the ultimate and deliberate expression of the author's views. It should be added, however, that in the majority of cases there are no important changes.) These volumes are for sale as follows, in paper covers:

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The Institute maintains at more than a hundred important mining centers throughout the world, free sets of its *Transactions*, open for consultation without fee, to all suitable applicants. Hence, the value of this index is by no means limited to individual possessors of complete sets of the *Transactions*. Moreover, the title of a paper, or the record of any remarks concerning a subject, being found in the Index, the Secretary's office of the Institute will supply upon written application any desired information as to the nature and length of said paper, whether it can be supplied in separate pamphlet form, etc.

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Special Editions.

"*The Genesis of Ore-Deposits*," comprising the famous treatise of the late Professor Franz Pošepný, with the successive discussions thereof by Le Conte, Blake, Winchell, Church, Emmons, Becker, Cazin, Rickard and Raymond (all of which were published in Volumes XXIII. and XXIV. of the *Transactions* of the Institute, and subsequently in the special "Pošepný Volume," issued by the Institute); also, later papers by Van Hise, Emmons, Weed, Lindgren, Vogt, Kemp, Blake, Rickard and others, and the discussions of these papers by De Launay, Beck, and many others (some of these were included in Volume XXX. and the remainder appeared in Volume XXXI.); also a complete bibliography of Institute papers and discussions on this subject from 1871 to the year 1902.

The original Pošepný volume comprised 265 pages, and was sold for $2.50, at which price the edition was long since exhausted. The present volume is an octavo of 825 pages.

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Spanish-American Mining and Metallurgical Glossary, bound in leather, pocket-size, 96 pages, ... 0.75

Year Book, containing List of Members, Rules, etc., paper, . 0.50

Pamphlets.

1. The Minutes of the Proceedings of each Meeting.

2. Such of the papers presented or read by title at each Meeting as are furnished by the authors and approved by the Council for full publication. (In nearly all cases in which papers, the titles of which appear in the Proceedings, are not subsequently published, they have been withdrawn by the authors.) These papers are published separately in pamphlet form, and are marked "subject to revision." Beyond the edition distributed in the *Bulletin*, without charge, to members and associates, a small supply is retained to meet subsequent demand. There are no copies on hand of papers read before 1880. The stock is nearly complete from 1880. These papers are for sale at the office of the Secretary, or are sent to purchasers, charges paid, on receipt of the price, as follows:

No. of Pages.	Single Copies.	10 Copies.	20 Copies.
24 or less	$0.25	$2.00	$3.50
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97 to 128	0.50	3.75	6.25
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Authors' Edition of Pamphlets.

Extra copies of pamphlets, if ordered before the printing of the *Bulletin*, will be furnished to members of the Institute at special rates, which will be given on application to the Assistant Secretary, Joseph Struthers, 29 West 39th St., New York, N. Y.

CONSTITUTION.

[Adopted Feb. 21, 1905.]

ARTICLE I.

Name and Object.

Sec 1. This Institute is incorporated under the Membership Corporation Law of the State of New York; its corporate name is American Institute of Mining Engineers; and its objects are such as are stated in its Certificate of Incorporation.

ARTICLE II.

Members.

Sec. 1. The membership of the Institute shall comprise four classes, namely: (1) Members; (2) Honorary Members; (3) Associates; and (4) Honorary Associates. Only Members and Associates residing within the United States of America, Republic of Mexico and Dominion of Canada shall be entitled to vote at the meetings of the Institute.

Sec. 2. All Members, Honorary Members, Associates and Honorary Associates of the American Institute of Mining Engineers as the same existed on the day of the incorporation of this Institute, are Members, Honorary Members, Associates and Honorary Associates, respectively, of this Corporation.

Sec. 3. The following classes of persons shall be eligible for membership in the Institute, namely: as Members and Honorary Members, all professional mining engineers, geologists, metallurgists or chemists, and all persons practically engaged in mining, metallurgy or metallurgical engineering; as Associates and Honorary Associates, all persons desirous of being connected with the Institute who, in the opinion of the Council, are suitable.

Sec. 4. Every candidate for election as a Member or Associate of the Institute must be proposed for election by at least three Members or Associates; must be approved by the Committee on Membership, as prescribed in the By-Laws; and must be elected by the Council. Not less than three-fourths of the votes cast shall be necessary to an election. Every person so elected shall become a Member or Associate, as the case may be, upon payment of his first dues as hereinafter prescribed. Each candidate for Honorary Member or Honorary Associate, must be recommended by at least ten Members or Associates; must be approved by the Council; and must be elected by ballot at a meeting of the Board of Directors by the unanimous vote of all the Directors present; provided, however, that the number of Honorary Members and Honorary Associates shall not at any time exceed twenty.

Sec. 5. If any person elected a Member or Associate does not, within sixty days after notice of his election, accept the same and pay his initiation fee and dues for the current year, his election may be cancelled at the discretion of the Council.

SEC. 6. The Council may at any time change the classification of a person elected as an Associate so as to make him a Member, or vice versa. All Members and Associates shall be equally entitled to the privileges of membership, provided that Honorary Members, Honorary Associates, and Members and Associates whose Post-Office addresses shall be outside of the United States, Mexico and Canada, shall not be entitled to vote.

ARTICLE III.

DUES.

SEC. 1. The dues of Members and Associates shall be Ten Dollars per annum, payable in advance on the first day of each Calendar year. Each newly elected Member or Associate shall pay, when notified of election, an initiation fee of Ten Dollars in addition to the dues for the current year. Honorary Members and Honorary Associates shall not be liable to initiation fee or dues. Any Member or Associate in arrears for one year may, at the discretion of the Council, be deprived of the receipt of publications or stricken from the list of Members, provided that he may be restored to membership by the Council on payment of all arrears or may be again proposed and elected after an interval of three years.

SEC. 2. Any Member or Associate not in arrears may become, by the payment of One Hundred and Fifty Dollars at one time, a Life Member or Associate; and shall not be liable thereafter to annual dues.

ARTICLE IV.

BUSINESS MEETINGS OF THE INSTITUTE.

SEC. 1. The annual meeting of the Institute for the election of Directors and transaction of other business shall take place on the third Tuesday in February in each year. A report of the financial condition of the Institute and an abstract of the accounts shall be furnished by the Directors, and presented at each annual meeting.

SEC. 2. Special business meetings of the Institute may be held at such times and places as the Board of Directors may appoint, upon notice to all Members and Associates entitled to vote, directed to each at his last known Post-Office address, and mailed in the City of New York not less than twenty days before the date fixed for such meeting.

SEC. 3. At all business meetings of the Institute the presence of nine Members and Associates shall constitute a quorum.

SEC. 4. At all business meetings of the Institute Members and Associates may vote either in person or by proxy, but no Member or Associate in arrears since the last annual meeting shall be entitled to vote.

ARTICLE V.

OTHER MEETINGS OF THE INSTITUTE.

SEC. 1. All meetings of the Institute other than business meetings shall be held at such times and places as the Council may appoint. Notice of all such meetings shall be given to all Members and Associates by mail.

ARTICLE VI.

Directors and Officers.

Sec. 1. The business and financial affairs of the Institute shall be managed by a Board of Directors, who shall be elected at the annual meeting in the manner prescribed in the Certificate of Incorporation.

Sec. 2. The officers of the corporation shall be a President, Vice-President, Secretary and Treasurer, who shall be elected by the Directors from among their number. All such officers shall be elected at the first meeting of the Board of Directors after each annual meeting of the corporation, and shall hold office for one year or until their successors are elected and qualify.

The duties of all officers shall be such as usually pertain to their offices, respectively, together with such other duties as may from time to time be prescribed for them by the By-Laws. The Treasurer shall give a bond for the faithful performance of his duties in a sum to be fixed by the Board of Directors, but at the expense of the Institute.

Sec. 3. In the event of a vacancy occurring in the Board of Directors by death, resignation or otherwise, the remaining members of the Board may, by a majority vote, elect a successor to fill the vacancy, who shall continue in office until the next annual meeting or until his successor shall have been chosen.

Sec. 4. The Board of Directors may, in its discretion, declare the place of any Director vacant, on his failure for any reason, to attend three successive meetings of the Board. Any Director who shall under this section or in any other manner cease to be a member of the Board shall, at the same time, be held to have vacated any other office to which he shall previously have been elected; and the Board shall elect a new incumbent to the said vacant office.

Sec. 5. The Board of Directors may from time to time appoint from their own number standing and special committees, and may delegate to such committees such duties as they may see fit.

ARTICLE VII.

Meetings of the Board of Directors.

Sec. 1. A regular meeting of the Board of Directors for the election of officers and the transaction of other business shall be held on the third Tuesday in February in each year, after the adjournment of the annual meeting of the Institute.

Sec. 2. Special meetings of the Board of Directors, at which any business may be transacted, may be called to meet at any time at the office of the Institute in the City of New York, by notice in writing mailed at least five days before the meeting, by the Secretary to each member of the Board at his last known Post-Office address, signed either by the President or the Vice-President or by three members of the Board.

Sec. 3. At all meetings of the Board of Directors the presence of five members shall constitute a quorum.

ARTICLE VIII.

The Council.

Sec. 1. The professional, technical, scientific and social interests of the Institute shall be committed to the supervision of a Council composed of a President

of the Council, six Vice-Presidents of the Council, a Secretary of the Council and nine Councilors, who shall be elected from among the Members and Associates of the Institute in the manner hereinafter prescribed. Members of the Council may or may not be members of the Board of Directors.

SEC. 2. The President of the Council shall be elected for one year, and no person shall be eligible for immediate re-election to this office who shall have held the same for two consecutive years.

After the first year Vice-Presidents of the Council shall be elected to serve for two years, and Councilors shall be elected to serve for three years. No Vice-President of the Council or Councilor shall be eligible for immediate re-election to the same office at the expiration of the term for which he was elected. The Secretary of the Council shall be elected annually.

SEC. 3. At the first annual meeting, to be held in the year 1905, there shall be elected a President of the Council to serve for one year, a Secretary of the Council to serve for one year, three Vice-Presidents of the Council to serve for one year, three Vice-Presidents of the Council to serve for two years, three Councilors to serve for one year, three Councilors to serve for two years, and three Councilors to serve for three years. At each subsequent annual meeting there shall be elected a President of the Council to serve for one year; a Secretary of the Council to serve for one year; three Vice-Presidents of the Council to serve for two years; and three Councilors to serve for three years. The term of office of all Members of the Council shall continue until the adjournment of the meeting at which their successors are elected.

SEC. 4. Vacancies in the Council may occur by death or resignation; or the Council may, by the vote of a majority of all its members, declare the place of any officer or member of the Council vacant, on his failure for one year, from inability or otherwise, to attend the regular meetings or perform the duties of his office. All vacancies shall be filled by the appointment of the Council, and any person so appointed shall hold office for the remainder of the term for which his predecessor was elected or appointed; *provided* that the said appointment shall not render such person ineligible for election to the Council at the next meeting.

SEC. 5. The presence of five members of the Council shall constitute a quorum; but the Council may appoint an Executive Committee, or any business coming within the authority of the Council may be transacted at a regularly-called meeting thereof, at which less than a quorum may be present, subject to the approval of a majority of the Council subsequently given in writing to the Secretary and recorded by him with the minutes.

SEC. 6. The election of the Council shall take place at the regular annual meeting of the Institute. Nominations for members of the Council may be sent in writing to the Secretary accompanied with the names of the proposers at any time not less than thirty days before the annual meeting; and the Secretary shall, not less than two weeks before said meeting, mail to every Member or Associate entitled to vote a list of all nominations for each office so received, together with the names of the persons ineligible for election to each office; and if the Council or a Committee thereof, appointed for the purpose, shall have recommended any nomination, such recommendation may also be sent to the Members and Associates with the list of all nominations made.

ARTICLE IX.

Meetings of the Council.

Sec. 1. Meetings of the Council shall be held at such times and places as the President of the Council or one of the Vice-Presidents of the Council may appoint.

Sec. 2. A meeting of the Council may be held on the day of the annual meeting of the Institute without previous notice. Written notice of all other meetings of the Council, specifying the time and place of such meeting, signed by the Secretary, shall be mailed to every member of the Council at his last known Post-Office address at least ten days before the date of the meeting.

ARTICLE X.

Papers and Publications.

Sec. 1. The Council shall have power to decide as to the acceptance and publication of any professional papers presented to the Institute, subject to such conditions as the Board of Directors may prescribe.

Sec. 2. The copyright of all professional papers communicated to and accepted by the Institute shall be vested in it, unless otherwise expressly agreed between the Council and the author. The Institute shall not assume responsibility for any statements of fact or opinion advanced in the papers or discussions at its meetings. Neither the Council nor the Institute shall officially approve or disapprove any technical or scientific opinion or any proposed enterprise, outside of the management of the meetings, discussions and publications of the Institute, and the conduct of its business affairs by the Board of Directors.

Sec. 3. Special Committees may from time to time be appointed by the Council to make investigations and prepare reports for presentation to the Institute, but no action shall be taken binding the Institute for or against the conclusions embodied in any such reports.

ARTICLE XI.

Suspensions and Expulsions.

Sec. 1. Any member of the Institute who shall be convicted of a crime involving, in the opinion of the Board of Directors, moral turpitude, shall, upon the passage by the Board of Directors of a resolution declaring the crime for which he has been convicted to be of such character, be thereupon dropped from membership in this Institute.

Sec. 2. Any member of the Institute may be suspended or expelled for misconduct by the Board of Directors, after charges setting forth such misconduct shall have been prepared by the Council and filed in writing with the Board. Upon the receipt of such charges in writing, the Board may, in its discretion, suspend such member pending a hearing and determination thereupon. As soon as may be after the receipt of such charges, the Board shall fix a date for a hearing thereupon and shall give to the accused member notice thereof in writing, mailed to him at his last known Post-Office address not less than thirty days before said date, accompanied by a full copy of the charges and a copy of the second, third and fourth sections of this article.

Sec. 3. Upon the day fixed for the hearing, the accused member may appear before the Board, either in person or by an accredited representative; hear any

witnesses who may be called in support of the charges and at his option cross-examine the same; and hear read any documentary evidence offered in support of the charges. The accused may, in his discretion, produce and examine witnesses in his defence, and submit documentary evidence, including a statement from himself in writing. After the conclusion of the hearing, the Board of Directors shall consider and vote to approve or disapprove the charges. If the Board shall, by a vote of two-thirds of its members, declare the charges sustained, it may suspend the member for a stated period or expel him.

Sec. 4. If the accused member shall not appear at the hearing, and shall within three months thereafter file with the Board an affidavit stating that he had not received notice of the charges against him in time to enable him to present his defence, the Board shall fix a date for a re-hearing within three months from the receipt of such affidavit and shall immediately notify the accused member by mail of such date. Upon the re-hearing, the accused shall have the same privilege of presenting his defence as he would have had upon the original hearing; and after the defence is presented, the Board shall take a new vote upon the charges, the result of which shall be conclusive.

Sec. 5. All interests in the property of the Institute of persons resigning, or otherwise ceasing to be Members or Associates, shall vest in the Institute.

ARTICLE XII.

Amendments.

Sec. 1. This Constitution or any Article or Section thereof may be amended at any annual meeting by a two-thirds vote of all the members present in person or by proxy, *provided* that notice of the proposed amendment shall have been given in writing at a previous meeting, and *provided also* that the amendment or amendments so adopted shall have been printed and mailed to all Members and Associates not later than thirty days before the annual meeting. Any amendment or amendments approved by a majority of the votes cast shall be deemed to have been adopted, and shall become a part of this Constitution. The Secretary shall forthwith print and distribute to Members and Associates an announcement of the result of said vote, and if any amendment or amendments shall have been adopted, a copy of the section or sections so amended.

BY-LAWS.

[Adopted Feb. 21, 1905. Amended Feb. 20, 1906, Nov. 16, 1906.]

I. Presiding Officers.

At all Business meetings of the Institute the President, or, in his absence, the Vice-President, or, in the absence of both of them, any other member of the Board of Directors to be chosen by the meeting, shall preside.

At all other meetings of the Institute the President of the Council or, in his absence, one of the Vice-Presidents, if present, shall preside.

II. Order of Business.

At each Business meeting of the Institute the order of business shall be as follows:

1. Reading of minutes of preceding meeting.
2. Report of the President.
3. Report of the Treasurer.
4. Report of the Secretary.
5. Election of Directors.
6. Election of Members of the Council.
7. Reports of Standing Committees.
8. Reports of Special Committees.
9. Special Orders.
10. Miscellaneous business.

This order of business may be changed by a vote of a majority of the Members and Associates present in person or by proxy.

The usual parliamentary rules shall govern all meetings of the Institute except in cases otherwise provided by the Constitution or the By-Laws.

At all sessions of the Institute other than business meetings, the order of proceedings and the time of adjournment shall rest in the discretion of the presiding officer.

III. Secretary.

The Secretary shall keep a record of the proceedings of all meetings of the Institute. He shall be custodian of the Corporate Seal, of the Minute Books, and of all Legal Documents belonging to the Institute. He shall conduct, on behalf of the Institute, all correspondence relating to business matters, except such as pertains directly to the office of the Treasurer.

He shall notify all officers and Directors and Members of the Council, and all Members of Committees of their election and appointment; shall issue notices of all meetings of the Board, and of the annual and other meetings of the Institute; and shall, in calling special meetings of the Directors, specify the object of such meeting.

IV. Secretary of the Council.

The Secretary of the Council shall act as the Clerk of that body at all of its meetings and at all meetings of the Institute called for the discussion of professional, technical or scientific matters, or for any other purpose than the transaction of business.

He shall be custodian of all technical or scientific papers submitted to the In-

stitute for its consideration, shall have charge of the editing and printing of all material published by the Institute, and of the distribution thereof. On the first day of May following the year in which each volume of *Transactions* is printed, he shall turn over to the Library Committee all copies of the same not theretofore distributed by him. He shall have charge of all the correspondence of the Institute relating to other than business affairs.

The Secretary of the Council shall receive a salary to be fixed by the Board of Directors. He may appoint an Assistant with the title of Editor, who shall likewise receive a salary to be fixed by the Board of Directors.

The Secretary of the Council may or may not be the same person as the Secretary of the Institute.

V. Assistant Secretary.

The Secretary may, with the approval of the Board of Directors, appoint an Assistant to whom both he and the Secretary of the Council may delegate such of his or their duties as he or they may see fit. This Assistant Secretary shall receive such salary as shall be fixed by the Board of Directors, which shall cover his services both to the Secretary and to the Secretary of the Council.

VI. Treasurer.

The Treasurer shall collect and, under the direction of the Board of Directors, shall disburse all funds of the Institute. He shall keep regular accounts in books belonging to the Institute, which shall be open to any member of the Board of Directors. He shall report in writing at each annual meeting of the Institute and at every meeting of the Board of Directors at which such report shall be called for, the balance of money on hand, and any existing appropriation which may affect the same.

His accounts shall be audited annually by a Committee of three Members or Associates to be appointed by the President at least thirty days prior to the annual meeting in each year, which Committee shall report thereon at such annual meeting.

The Treasurer may, at his discretion, place funds of the Institute, not at any time exceeding $5,000, in a special account in a Bank or Trust Company, subject to the draft of the Assistant Treasurer, and may delegate to the Assistant Treasurer the duty of paying, out of this account, the current expenses of the Institute.

The Treasurer shall be solely responsible to the Institute for all moneys received, whether the same are entrusted to the Assistant Treasurer or not.

VII. Assistant Treasurer.

The Treasurer may appoint, with the approval of the Board of Directors, an Assistant Treasurer, to whom he may delegate the duty of conducting the correspondence incidental to the office of Treasurer, of receiving and depositing in bank to the credit of the Institute all moneys received, and of paying, out of the special account upon which he may be authorized to draw, the necessary expenses of the Institute. The Treasurer may require of him a bond, running to the Treasurer personally, in an amount not exceeding $5,000, the expense of which shall be borne by the Institute.

The Assistant Treasurer shall receive such compensation as shall be fixed by the Board of Directors.

The offices of the Assistant Secretary and of the Assistant Treasurer may, if

so desired by both the Secretary and the Treasurer and approved by the Board of Directors, be united in the same person, who shall then receive the salary of both offices.

The Assistant Treasurer may, with the approval of the Board of Directors, employ such persons as are necessary to constitute a clerical and office force for himself, the Assistant Secretary and the Secretary of the Council, at such salaries as shall be approved by the Board of Directors. He shall, if the offices of Assistant Secretary and Assistant Treasurer be united in the same person, be the immediate superior of all such employees, unless the Secretary of the Council or the Treasurer be present, in which event either of them shall be the superior of all employees, including their respective assistants.

VIII. Standing Committees.

The Standing Committees of the Institute shall be three in number, known respectively as the Finance Committee, the Library Committee and the Committee on Membership.

The Finance Committee and the Library Committee shall each consist of three members of the Board of Directors, and shall be appointed by the President at the first meeting of the Board, after the annual meeting in each year.

The Committee on Membership shall consist of five Members of the Council, and shall be appointed by the President of the Council, at the first meeting of the Council after the first annual meeting in each year.

IX. Finance Committee.

It shall be the duty of the Finance Committee to inquire into and examine the financial condition of the Institute, and to consider ways and means of increasing its revenues and of limiting its expenses. It shall report from time to time to the Board as often as it may deem expedient, and whenever it shall be directed so to do; and the Treasurer shall at all times furnish it with such statements and information as it may desire.

It shall determine the investment of such surplus moneys as shall from time to time accrue to the Institute. It shall, at least once in each year, examine the securities belonging to the Institute in the custody of the Treasurer, and report thereon to the Board.

It may, at any time, examine the books and vouchers of the Treasurer and Assistant Treasurer.

The Treasurer shall not be a member of the Finance Committee, but shall attend the meetings of the same if requested to do so.

X. Library Committee.

The Library Committee shall be the custodian of all books in the Institute Library and of additions thereto; also of all back numbers of the *Transactions* of the Institute. It shall, on the first day of May, of each year, receive from the Secretary of the Council, and receipt for same to him, all the volumes of *Transactions* for the preceding year, not then distributed by said Secretary.

It shall cause to be kept, under the direction of the Assistant Secretary, a catalogue of all books in the Library and an account in ledger form of all volumes of *Transactions* in its custody, in which shall be charged to it all volumes delivered to it, and in which shall be credited all volumes taken from its custody for sale or for any other purpose.

The receipts from the sale of any volume of *Transactions* taken from the custody of the LIBRARY COMMITTEE shall be credited to the LIBRARY COMMITTEE on the books of the Treasurer, and devoted to the general purposes of the Institute.

XI. COMMITTEE ON MEMBERSHIP.

All nominations for Members or Associates of the Institute shall be submitted to and passed upon by the COMMITTEE ON MEMBERSHIP, who shall report thereon to the Council. It shall receive and consider all communications respecting candidates, and shall make diligent inquiry as to the character and qualifications of each one. Its proceedings shall be secret and confidential.

No member of the Committee shall propose any candidate.

XII. ELECTION OF MEMBERS.

After the COMMITTEE ON MEMBERSHIP shall have reported to the Council its conclusions as to the acceptability of each candidate, the Council shall vote upon the same.

Two negative votes of members of the Council present shall prevent the election of any candidate. No person shall be proposed for election to the Institute within one year after his name shall have been rejected by the Council.

XIII. UNITED ENGINEERING SOCIETY.

The Board of Directors shall, at its first meeting after the adoption of these By-Laws, designate three Members or Associates of this Institute to be representatives of this Institute upon the Board of Trustees of the UNITED ENGINEERING SOCIETY, making at the same time provision for the expiration of the terms of office of said representatives, as provided in the By-Laws of the said UNITED ENGINEERING SOCIETY.

At the last meeting of the Board of Directors prior to the first day of each January thereafter, the Board shall designate a Member or Associate of this Institute to be a representative of this Institute upon the Board of Trustees of the said UNITED ENGINEERING SOCIETY for a period of three years beginning at the next ensuing annual meeting of said Society.

At any time when a vacancy shall occur in the representation of this Institute in the Board of Trustees of said Society, by reason of the death, resignation or removal of any such representative therein, the Board of Directors of this Institute shall designate a Member or Associate to fill such unexpired term.

XIV. PUBLICATIONS.

The publications of the Institute shall include a periodical, called the *Bi-Monthly Bulletin* of the American Institute of Mining Engineers, which shall contain reports of proceedings, professional papers, notices, and other matter of interest to members. From the annual dues paid by each Member or Associate, five dollars shall be deducted and applied as a subscription to the *Bi-Monthly Bulletin* for the year covered by such payment.

XV. AMENDMENTS.

These By-Laws may at any time be altered or amended by a vote of two-thirds of the Board of Directors, or by the Members, at a business meeting of the Institute, in the same manner provided for amendments of the Constitution in Article XII. thereof.

ANNUAL MEETING OF THE INSTITUTE.

At the Annual Business Meeting of the Institute, held Feb. 18, 1908, the following persons were elected by vote of the members and associates in person or by proxy:

Council.

President.

(To serve for one year.)

John Hays Hammond,	New York, N. Y.

Vice-Presidents.

(To serve for two years.)

J. Parke Channing,	New York, N. Y.
F. W. Denton,	Painesdale, Mich.
John B. Farish,	Denver, Colo.

Secretary.

(To serve for one year.)

R. W. Raymond,	New York, N. Y.

Councilors.

(To serve for three years.)

C. R. Corning,	New York, N. Y.
R. V. Norris,	Wilkes-Barre, Pa.
W. H. Shockley,	Tonopah, Nev.

Directors.

(To serve for three years.)

James Douglas,	New York, N. Y.
James F. Kemp,	New York, N. Y.
Albert R. Ledoux,	New York, N. Y.

[Secretary's Note.—The complete list of all officers of the Institute will be found on p. viii. of this volume. The following explanation, first published in *Bi-Monthly Bulletin*, No. 8, March, 1906, p. viii., is here repeated in order to recall to old members, and convey to new ones, the relations of the two governing bodies as determined by the Certificate of Incorporation of the Institute, and the Constitution and By-Laws adopted in accordance therewith.

The body legally responsible for the business management is the Board of nine Directors (three elected annually to serve three years), which elects its own officers. This body, for reasons of practical convenience, is composed of well-known members residing in New York City, and able to attend, without serious inconvenience or expense, the necessary meetings of the Board. The officers of this Board are legally the officers of the Institute. But, apart from business management, the Board

exercises no control over the election of members, or the professional and technical work of the Institute, except that its vote is required to elect honorary members, upon the recommendation of the Council.

The Council is a body constituted in all respects (except that it has no Treasurer) like the Council existing before the incorporation of the Institute, in January, 1905, and charged with all duties and powers, except those which the Board of Directors must legally perform. It elects members, appoints the times and places of professional meetings, and controls the publication and distribution of papers and volumes, etc. Its members (President, Vice-Presidents and Councilors) are elected by the members of the Institute, voting in person or by proxy, and after publication of the nominations received; and it is intended to represent, as far as practicable, both the professional and the geographical distribution of the membership. Consequently, whatever professional honor attaches to official position belongs to membership in the Council, rather than in the legal Board of Directors. This remark implies no disparagement of the members of the latter body, every one of whom has served, or is now serving, as a member of the Council. But it is only fair to explain that their election and continued re-election as Directors is simply a matter of legal convenience.]

Proposed Amendments to the Constitution.

Notice was given in writing, as required by Article XII. of the Constitution, of the following amendments, proposed for consideration at a future business meeting:

To Art. II. After the first sentence of this Article, add the following sentence: "These classes may be sub-divided by the Council, according to profession, length of membership, nationality, or other conditions not inconsistent with the provisions of this article."

To Art. III. In the first line, substitute "fifteen" for "ten" dollars.

PROCEEDINGS OF THE BOARD OF DIRECTORS.

The following acts of the Directors are reported for the information of members:

At a meeting held June 20, 1907, Dr. Charles D. Walcott, Director of the U. S. Geological Survey, Washington, D. C., having been recommended by unanimous vote of the Council, was unanimously elected an Honorary member of the American Institute of Mining Engineers, in recognition of his personal services to American geology, and of his generous co-operation with this Institute, to the *Transactions* of which he has not only himself made valuable contributions, but has also encouraged the preparation of such contributions by the members of the U. S. Geological Survey under his charge, his liberal policy in this respect having gone so far as to permit the preliminary publication by the Institute of important results reached by the geologists of that Survey, even before their final and full publication as public documents. This wise and generous course has not only brought the U. S. Geological Survey into intimate and sympathetic relations with the mining-industry of the world, but has also enriched the *Transactions* of the Institute, especially in the department of the science of ore-deposits, with many valuable papers.

At a meeting held Jan. 14, 1908, Mr. Charles Kirchhoff was unanimously re-elected a Trustee of the United Engineering Society, to serve for a second term of three years.

At a meeting held Feb. 18, 1908, directly after the adjournment of the annual meeting of the Institute, the following officers were elected for the ensuing year: *President*, James Gayley; *Vice-President*, James Douglas; *Secretary*, R. W. Raymond; *Treasurer*, Frank Lyman.

Financial Statement.

The following statement of receipts and expenditures from Jan. 1 to Dec. 31, 1907, is published by authority of the Board of Directors:

RECEIPTS.

Balance from statement of January, 1907,		$1,238.78
Annual dues,*	$37,483.85	
Life memberships,	2,460.00	
Initiation fees,	3,010.54	
Binding of *Transactions*,	3,585.65	
Sale of publications,	3,504.35	
Electrotypes,	35.90	
Miscellaneous receipts,	330.71	
Advertising,	3,536.47	
		53,947.47
Interest on bonds and deposits,		1,132.68
Reimbursement from Special Fund for balance of installments of principal and part of interest paid to the United Engineering Society on mortgage in 1906, now reimbursed,		13,400.00
Reimbursement from Library Fund for library additions (1907),		218.67
		$69,937.60

DISBURSEMENTS.

Printing Vol. XXXVII. of the *Transactions*, *Bi-Monthly Bulletin* and extra pamphlets,	$11,567.18	
Printing circulars and ballots,	131.25	
Binding Vol. XXXVII. of the *Transactions*,	3,287.27	
Binding miscellaneous volumes,	425.33	
Binding of exchanges,	300.00	
Engraving and electrotyping,	716.85	
Secretary's department, including clerks, stenographers, and expenses of editing and proof-reading,	10,219.29	
Treasurer's department, including collection of dues, shipping, etc.,	5,629.63	
Librarian and assistants,	1,534.00	
Postage,	3,721.57	
Stationery,	630.22	
Express and freight charges,	1,672.94	
Telephone,	241.66	
Telegrams, cables, carfares, etc.,	69.33	
Office supplies and repairs,	153.72	
Refunding over-payments,	39.61	
Insurance premiums,	246.43	
Collection charges,	36.83	
Extra clerical assistance,	156.75	
Special stenographers and expenses of meetings,	669.14	
Auditing,	125.00	
Advertising, expenses and commissions,	1,495.58	
Office cleaning and sundry expenses,	31.70	
Carried forward,		$43,101.28

* $17,470 of this amount has been applied to subscriptions to the *Bi-Monthly Bulletin* in accordance with post-office regulations.

Brought forward,		$43,101.28
Interest at 4 per cent. on $130,000 principal of land mortgage on 29 West 39th St.,	$5,200.00	
Quota of current expenses of building 29 West 39th St.,	10,000.00	
		15,200.00
Special editing, compiling, printing and binding special edition, Index Vols. I. to XXXV.,		3,625.90
Rent for part of offices, 99 John St., during January, February and March,	250.00	
Storage of *Transactions*,	129.45	
		379.45
Library additions (expenditure from appropriation of $500),		473.73
Office improvements, including shelving,		796.89
Furniture, etc., amount expended of appropriation of $2,000,		1,519.39
Sundry fittings and incidentals,		534.21
Balance,		4,306.75
		$69,937.60

New York, N. Y., February 4, 1908.

We have examined the above statement, compared it with the books and vouchers and find same correct.

(Signed) Barrow, Wade, Guthrie & Co.,
Certified Public Accountants.

REPORT OF THE COUNCIL FOR THE YEAR 1907.

Meetings.

Two meetings for the reading and discussion of papers, etc., have been held during the year 1907—namely, the Ninety-second meeting, held April 16 to 20 in New York, N. Y., and the Ninety-third meeting, held July 23 to 30 in Toronto, Canada.

The proceedings of these meetings, including descriptions of the entertainments and excursions connected therewith, have already been published and distributed to the members of the Institute; the New York meeting in *Bi-Monthly Bulletin*, No. 15, May, 1907, pp. 541 to 559, and the Toronto meeting in *Bi-Monthly Bulletin*, No. 17, September, 1907, pp. 847 to 875. In addition to these "Proceedings," there were also published in *Bi-Monthly Bulletin*, No. 15, May, 1907, an illustrated description of the United Engineering Society building (pp. vi. to xxvi.), and a record of the exercises attending its dedication, including the addresses of Andrew Carnegie, Arthur T. Hadley, Samuel Sheldon, Frederick R. Hutton, John Hays Hammond, and T. C. Martin (pp. xxxviii. to lxv.).

At the New York meeting, 43 papers and discussions were presented, and the names of 297 members and guests were registered at the Institute headquarters; this number, however, does not represent all who were present at the sessions and excursions. At the Toronto meeting, 23 papers and discussions were presented. The number of members and guests registered at the Toronto headquarters was 191, which doubtless does not include the names of all who were present at the sessions, or took part in the numerous excursions and visits around Cobalt.

Publications.

Transactions.—Volume XXXVII. of the *Transactions*, an octavo of 1,044 pages, comprising 51 papers and discussions presented during the year 1906, was issued and distributed early in July, 1907.

Bi-Monthly Bulletin.—Six numbers of the *Bi-Monthly Bulletin* (Nos. 13 to 18), containing the "subject to revision" edition of technical papers of the Institute and announcements of general interest to the members of the Institute, such as Library accessions and requirements, lists of new members and associates, lists of proposed members and associates, changes of address, deaths of members, progress on the United Engineering Society building, etc., have been published and distributed promptly throughout the year 1907. The number of pages occupied by technical papers, and "subject and author" index, amounts to 1,030, to which are to be added 252 pages of announcements, and 128 pages of advertising matter, making a total of 1,510 pages of printed matter.

Index.—A complete analytical and alphabetical Index of Volumes I. to XXXV. (inclusive) of the *Transactions* was issued and distributed to subscribers in November, 1907. The work consists of 700 pages, bound in cloth or half morocco. The magnitude of the Index, involving the collection, sorting, classification, and completion of many thousand titles, names and cross-references, can only be fully appreciated by those who have done work of similar character. Comparatively few complete sets of *Transactions* are in the possession of the members of this Institute, and the number of such complete sets remaining on hand is very small. The volumes have never been stereotyped, and it is not likely that any of them will be reprinted. The Institute maintains, at more than a hundred important mining-centers throughout the world, free sets of its *Transactions*, open for consultation, without fee, to all suitable applicants.

To those who do not possess a complete set of the *Transactions*, this Index is of special value, since it shows not only the contents of all the volumes, but also, from the table of contents, the papers contained in any one volume. Moreover, members not infrequently send for publication valuable professional papers in which previous contributions to the *Transactions*, directly or indirectly dealing with the same subjects, are ignored. The author of an Institute paper should recognize what his fellow-members have done before him in the same line, which he cannot do unless he is acquainted with the contents of the back volumes of the *Transactions*. In this connec-

tion, the Index will prove indispensable. Finally, the issue of this Index is part of the plan to render available to those members at a distance the privilege of consulting by correspondence books and periodicals in the combined libraries of the three founder societies (now exceeding 50,000 volumes of scientific and engineering works and 450 current technical journals and magazines), in order to obtain copies or abstracts of text and drawings.

The index-volume, of which an edition of 1,600 was printed, has not been stereotyped; the price of the cloth-bound copy is $5; of the half-morocco bound, $6.

Membership.

Changes in membership have taken place during the year as follows: 1 honorary member (previously on the regular list of members), 291 members and 12 associates have been elected; 1 member has been elected an honorary member and 12 associates have become members; the deaths of 44 members and 4 associates have been reported; 43 members and 7 associates have resigned; and 63 members and 4 associates have been dropped from the roll by reason of non-payment of dues, loss of correct address, etc.* These changes are shown in the accompanying table.

The total membership on January 1, 1908, was 4,192, as compared with 4,049 on January 1, 1907—a net gain for the year of 143 members.

Membership of the American Institute of Mining Engineers, Jan. 1, 1908.

	Honorary Members.	Members.	Associates.	Totals.
Membership Dec. 31, 1906	11	3,861	177	4,049
Gains: By Election	.....	291	12	303
Change of Status	1	12	.....	13
Reinstatement	.....	3	.....	3
Re-election	.....	2	.....	2
Losses: By Resignation	.....	43	7	50
Change of Status	.....	1	12	13
Dropping	.....	63	4	67
Death	.....	44	4	48
Total gains	1	308	12	321
Total losses	.....	151	27	178
Membership Dec. 31, 1907	12	4,018	162	4,192

* Many of these, no doubt, will be reinstated, as has been the case in former years.

The list of deaths reported during the year 1907 comprises the following names, the figures in parentheses indicating the year in which the persons named were elected to membership:

Members and Associates.—Thomas T. Baker (1887), Charles W. Benton (1897), Carl W. Bildt (1886), John Blatchford (1897), John Blue (1888), William R. Boggs, Jr. (1882), George L. Bradley (1874), T. Forster Brown (1886), E. E. Burlingame (1882), Chauncey E. Butler (1896), George V. Devinny (1903), George H. Evans (1898), J. K. Eveleth (1887), A. W. Fiero (1886), William W. Garrett (1893), William Glenn (1881), George B. Hanna (1887), B. J. Harrington (1877), Frank J. Hearne (1874), Christopher Henne (1900), George C. Hewett (1883), Charles J. Hillard (1888), Karl Howard (1898), Thomas J. Hurley (1899), H. E. Ingram (1894), Thomas E. Johns (1904), William J. Johnston (1901), James F. Jones (1888), Winfield S. Keyes (1872), Gustavus W. Lehmann (1892), Walter Leisenring (1891), William H. Long (1882), Frederick W. Mathews (1904), Robert S. Mercur (1900), Charles A. Molson (1887), William G. Neilson (1872), Louis Pelatan (1894), Alfred M. Rock (1904), George W. Rose (1904), Gilbert C. Simpson (1902), Thomas W. P. Storey (1906), Sidney Thow (1902), Henry T. Townsend (1879), Jean A. Variclé (1905), John A. Walker (1878), Eugene B. Willard (1900), Lewis Williams (1883), James W. R. Young (1891).

Of these, William G. Neilson has been made the subject of a special Biographical Notice. (See *Transactions*, xxxviii., Frontispiece, and pp. 402 to 411.) Other biographical notices have been published from time to time in the *Bi-Monthly Bulletin*, as given in the Index of this volume, under the name of the member.

MEMBERSHIP.

The following list comprises the names of those persons elected as members, who duly accepted election during the year 1908. The marks used to designate the different classes of membership are: Life Member, **; Member, *; Associate Member, †.

*Charles W. Abbott, ... Pioche, Nev.
*Louis K. Acker, Jr., ... Pittsburg, Pa.
*Guillermo A. Alamos, ... Chillan, Chile.
*James S. Anderson, ... Eccles, Manchester, Eng.
*William Aplin, ... Moonta, South Australia.
*James Archbald, Jr., ... Scranton, Pa.
*Harry J. Arkell, ... Cape Gracias, Nicaragua.
*Albert H. B. Arnold, ... Joplin, Mo.
*Wallace L. Atkinson, ... Treadwell, Alaska.
*Edwin A. Austin, ... Stockton, Cal.
*Augustus E. Bachert, ... Robertsdale, Pa.
*Luther W. Bahney, ... Stanford University, Cal.
*William H. Bainbridge, ... Santa Monica, Cal.
*C. J. Seymour Baker, ... London, Eng.
*Hugh C. Baker, ... North Tonawanda, N. Y.
*Harry P. Banks, ... Chicago, Ill.
*Tracy Bartholomew, ... Denver, Colo.
*Harvey Bassler, ... Myerstown, Pa.
**Horace A. Beale, Jr., ... Parkesburg, Pa.
*Charles P. Berkey, ... New York, N. Y.
*J. Ferguson Black, ... Sudbury, Ontario, Can.
*D. E. Blake, ... Birmingham, Ala.
*Richard F. Bohler, ... Vienna, Austria.
*Albert J. Bone, ... Copperhill, Tenn.
*William L. Borthwick, ... Tenabo, Nev.
*Corey C. Brayton, ... Asientos, Aguascalientes, Mex.
*Jules R. Breuchaud, ... Santa Rosa, Guanajuato, Mex.
†Carl Brewer, ... Ishpeming, Mich.
*Paul K. Brill, ... Guanajuato, Mex.
*George S. Brooks, ... Depue, Ill.
*Herbert O. Brooks, ... Albuquerque, N. M.
*Alfred Brown, ... Wallaroo, South Australia.
*Robert L. Brown, ... Auburn, Ala.
*William N. Brown, ... Gainesville, Va.
*Robert A. Bryce, ... Cobalt, Ontario, Can.
*Walter C. Buchanan, ... San Luis de la Paz, Guanajuato, Mex.
*Lloyd T. Buell, ... Copperhill, Tenn.
*Laurence N. B. Bullock, ... Copala, Sinaloa, Mex.
*Otto C. Burkhart, ... Blacksburg, Va.

*Clifford D. Caldwell, ... Pennington Gap, Va.
*Forest B. Caldwell, ... Santa Lucia, Sinaloa, Mex.
*Willis S. Caypless, ... Denver, Colo.
†R. Gordon Chaney, Jr., ... Colorado Springs, Colo.
*Fred C. Christy, ... Phœnix, Ariz.
*Hyacinthus P. Civretto, ... Unsan, Korea.
*Horace H. Clark, ... Chicago, Ill.
†John M. Clark, ... Toronto, Can.
*William B. Clark, ... Baltimore, Md.
*Clare L. Colburn, ... Denver, Colo.
*H. McKean Conner, ... Prince, W. Va.
*C. V. Corless, ... Victoria Mines, Ontario, Can.
*Edmund J. D. Coxe, ... Houtzdale, Pa.
*Charles N. Crary, ... Valdez, Alaska.
*Murray F. Crossette, ... Oracle, Ariz.
*Ignacio Diaz-Ossa, ... Santiago, Chile.
*James C. Dick, ... Salt Lake City, Utah.
*George H. Dickson, ... Lethbridge, Alberta, Can.
*John F. Dickson, ... Houston, Tex.
*Edgar V. Dodge, ... San Diego, Cal.
*John Henry Dowe, ... London, Eng.
*Augustus W. Drake, ... Lattimer Mines, Luzerne Co., Pa.
*Clarence G. Dresser, ... Goldfield, Nev.
*Harry C. Dudley, ... Marble, Minn.
*Morse S. Duffield, ... Salt Lake City, Utah.
*Alexander W. Edelen, ... Bonanza, Zacatecas, Mex.
†John H. Eggers, Jr., ... Alameda, Cal.
†Joseph B. Elwell, ... New York, N. Y.
*Carolus D. Emmons, ... Eugene, Ore.
*Raymond H. Emtage, ... Watkins, Barbados, West Indies.
*Joseph F. Erdlets, Jr., ... Salt Lake City, Utah.
*George W. Evans, ... Seattle, Wash.
†Charles G. Ewing, ... St. Louis, Mo.
*Samuel C. Faneuf, ... Rosario, Sinaloa, Mex.
*John H. Farrell, ... Marquette, Mich.
*Erminio Ferraris, ... Monteponi, Sardinia, Italy.
*Howell Fisher, ... Boston, Mass.
*Charles Fowles, ... Boston, Mass.
*Arthur C. Fox, ... Virginia, Minn.
*Joseph H. Frantz, ... Columbus, Ohio.
*Porter T. Frizzell, ... Portland, Ore.
*Chester A. Fulton, ... Guanajuato, Mex.
*Rudolf Gahl, ... Morenci, Ariz.
*Henry S. Geismer, ... Birmingham, Ala.
*John C. Gibson, ... Duquoin, Ill.
†Frank A. Gleason, ... Scranton, Pa.
*William H. Goodchild, ... London, Eng.
*Boris Gorow, ... Taxco, Guerrero, Mex.
*Chester H. Graves, ... Cedartown, Ga.
*Thomas B. Greenfield, ... Akmolinsk, Siberia.
*Adolph J. Greit, ... New York, N. Y.
*John M. Grice, ... El Oro, Mexico, Mex.

*William V. Griffith, ... La Fundicion, Peru.
*Heinrich F. Grondijs, ... Oruro, Bolivia.
*William Gummers, ... Roebling, N. J.
*Robert W. Hadden, ... Albuquerque, N. M.
*D. F. Haley, ... West Gore, N. S., Can.
*George T. Hansen, ... Milwaukee, Wis.
*William Harris, ... Seattle, Wash.
*Harry S. Harrop, ... Wilkinsburg, Pa.
†Tetsutaro Hasegawa, ... Yayesucho, Kojimachiku, Tokyo, Japan.
*H. B. Tancred Hawkins, ... Ballydehob, Ireland.
*Arthur E. Hepburn, ... Vancouver, B. C., Can.
*Donnel F. Hewett, ... Pittsburg, Pa.
*Joseph G. Hibbs, ... Philadelphia, Pa.
*Henry T. Hinrichsen, ... Lota, Chile.
*C. Barnes Hoadley, ... Searchlight, Nev.
*Samuel O. Hobart, ... Pottstown, Pa.
*George Hohagen, ... Potosi, Bolivia.
*Elmer A. Holbrook, ... Hedley, B. C., Can.
*Roy J. Holden, ... Blacksburg, Va.
*Manuel H. de Hora, ... Butte, Mont.
*Reginald E. Hore, ... Kingston, Ontario, Can.
*Frederick A. Horswill, ... Valley Springs, Cal.
*Frederick J. Horswill, ... Oakland, Cal.
*Cuthbert B. Horwood, ... Johannesburg, Transvaal, S. Africa.
*Arthur J. Hoskin, ... Golden, Colo.
*George A. Howells, ... New York, N. Y.
*Archibald J. Hunt, ... Ojuela, Mapimi, Durango, Mex.
*Thatcher R. Hunt, ... Iron Mountain, Cal.
*Clements F. V. Jackson, ... Brisbane, Queensland, Australia.
*Frederick H. Jackson, ... Pachuca, Hidalgo, Mex.
*George R. Jackson, ... Princeton, Mich.
*Lewis A. Jeffs, ... Salt Lake City, Utah.
*Weston Jenkins, Jr., ... Columbia, Pa.
*George Johnson, Jr., ... Ballarat, Cal.
*Kenji Kaishima, ... Penarth, Wales.
*Arthur L. Kelley, ... Tonopah, Nev.
*Edward F. Kenney, ... Johnstown, Pa.
*Martin H. Kilgour, ... New York, N. Y.
*Harold F. King, ... Kingston, N. Y.
*Bethune G. Klugh, ... Toledo, O.
*Shun Tet Kong, ... Changsha, Hunan, China.
*John F. Kramer, ... Robesonia, Pa.
*Michael H. Kuryla, ... El Oro, Mexico, Mex.
*Joseph R. Lashbrooke, ... Mojave, Cal.
*Frederick E. Laube, ... Treadwell, Alaska.
*Robert B. Law, ... Bangkok, Siam.
*Willis Lawrence, ... Yuma, Ariz.
*John E. Leibfried, ... Chicago, Ill.
*Arthur H. Lewis, ... Hazleton, Pa.
*Henry M. Lewis, Jr., ... Hartshorne, Okla.
*Julius A. Lewisohn, ... New York, N. Y.
*Frank A. Linforth, ... Butte, Mont.

*Duncan F. McAulay, ... Cobar, N. S. W., Australia.
*Ross D. McCausland, ... Chihuahua, Chihuahua, Mex.
*Charles R. McCollom, ... Calumet, Mich.
*Frederick F. McIntosh, ... Glen Osborne, Pa.
*John W. McKim, ... Salt Lake City, Utah.
*Alexander C. S. McLeod, ... Daylesford, Victoria, Australia.
*James H. McManaman, ... Careyville, Tenn.
*A. T. C. McMaster, ... Toronto, Can.
*Samuel D. McMiken, ... Komata, Auckland, N. Z.
*Anthony J. McMillan, ... Rossland, B. C., Can.
*William McMurtrie, ... New York, N. Y.
*Alfred M. Mackilligin, ... Horeham Road, Sussex, Eng.
*Prosper A. Maignen, ... Philadelphia, Pa.
†A. Bell Malcomson, Jr., ... Freeport, N. Y.
*Nathan H. Mannakee, ... Williamson, W. Va.
*Robert Marsh, Jr., ... Kimberly, Nev.
*Philip L. Marston, ... Tonopah, Nev.
*F. Oskar Martin, ... Mullan, Idaho.
*Robert L. Martin, Jr., ... Pittsburg, Pa.
*H. Maschmeyer, ... Hoboken-les-Anvers, Belgium.
*Thomas W. Mather, ... Guayaquil, Ecuador.
*Ellwood V. Matlack, ... St. Louis, Mo.
*F. Charles Merry, ... Ferguson, B. C., Can.
**George E. Montandon, ... Graneros, Chile.
*Robert B. Moran, ... Palo Alto, Cal.
*William J. Morrison, ... Toronto, Can.
*Martin Munroe, ... Murulidih, Bengal, India.
†Desaix B. Myers, ... Mine La Motte, Mo.
*Frederick B. Nold, ... Las Esperanzas, Coahuila, Mex.
*William G. Norrie, ... Coleman, Alberta, Can.
*George W. Otterson, ... Seattle, Wash.
*William D. Owens, ... Pittston, Pa.
*Frederic C. Paine, ... Goldfield, Nev.
*Harry G. Palsgrove, ... Denver, Colo.
*Henry M. Parks, ... Corvallis, Ore.
†Lewis A. Parsons, ... New York, N. Y.
*Henry H. Patterson, ... Newark, Cal.
*Hugh K. Picard, ... London, Eng.
*Robert S. Porter, ... Wilkes-Barre, Pa.
*De Villiers G. Pritchard, ... Cradock, Cape Colony, South Africa.
*James Pryor, ... Wallaroo, South Australia.
*Stephen E. Puckette, ... Darbyville, Va.
*Charles H. Purcell, ... La Fundicion, Peru.
*Francis R. Pyne, ... Great Falls, Mont.
*Bertram D. Quarrie, ... Cleveland, Ohio.
†James M. Raine, ... Meadow Creek, W. Va.
*William Ramsay, ... Knoxville, Tenn.
*M. N. Srinivas Rao, ... Bombay, India.
*Edwin C. Reeder, ... Houghton, Mich.
*Alexander S. Reid, ... London, Eng.
*Robert S. Reid, ... Sydney, N. S. W., Australia.
*José J. Reynoso, ... Mexico City, Mex.
*Eugene R. Rice, ... Wickenburg, Ariz.

*Samuel H. Richardson, Jr., ... Republic, Wash.
*Hallet R. Robbins, ... New York, N. Y.
*Edwin G. L. Roberts, ... Eltham, Kent, Eng.
*Herbert W. Ross, ... La Fundicion, Peru.
*Henry C. Schmidt, ... Monterey, N. L., Mex.
*Cyrus E. Scott, ... Fairmont, W. Va.
*Walter M. H. Scott, ... Smuggler, Colo.
*Alexander C. Scoular, ... Ellerslie, Whitehaven, Eng.
*Walter E. Segsworth, ... Toronto, Can.
*Lee H. Skeels, ... Urique, Chihuahua, Mex.
*Dale Slusher, ... Coalinga, Cal.
*Robert Smart, ... White Horse, Y. T., Can.
*Howard I. Smith, ... Wendendale, Ariz.
*Lloyd B. Smith, ... State College, Pa.
*John G. Smyth, ... Fairmont, W. Va.
*Jonathan P. Snow, ... Boston, Mass.
*Earle W. Sommercamp, ... Silver City, Idaho.
*Grant T. Stephenson, ... Wells, Mich.
*Douglas B. Sterrett, ... Washington, D. C.
*George W. Stubbs, ... Albuquerque, N. M.
*Charles C. Swartz, ... South Norwalk, Conn.
*Henry B. Taylor, ... Kansas City, Mo.
†John F. Thomas, ... Sharon, Pa.
*James S. Thompson, ... Stone Canyon, Cal.
*Benjamin F. Tibby, ... Salt Lake City, Utah.
*Thomas M. Topp, ... Turlock, Cal.
*Charles E. Turner, ... London, Eng.
*Nathaniel P. Turner, ... Firmeza, Santiago, Cuba.
*George B. Upton, ... Wickenburg, Ariz.
*Washington B. Vanderlip, ... New York, N. Y.
*Harry L. Venables, ... Lima, Peru.
*Robert E. B. Vinicombe, ... Akmolinsk, Siberia.
†Hugh Vivian, ... Caterham Valley, Sussex, Eng.
*Wilfrid B. Wainewright, ... Los Angeles, Cal.
*Edwin R. Wash, ... Pontiac, Ill.
*William J. Watson, ... Ladysmith, B. C., Can.
*Alois Weiskopf, ... Hanover-Linden, Germany.
*James S. C. Wells, ... New York, N. Y.
*James G. West, Jr., ... Braddock, Pa.
*William G. Whildin, ... Lansford, Pa.
*Edwin E. White, ... Fairmont, W. Va.
*Joseph E. White, ... Searchlight, Nev.
*W. P. White, ... Spokane, Wash.
†Louis H. Winkler, ... Johnstown, Pa.
*Job H. Winwood, ... Salt Lake City, Utah.
*Edwin F. Yates, ... Rawhide, Nev.
*Howard P. Zeller, ... Las Esperanzas, Coahuila, Mex.

Deaths.

The following list comprises the names of members whose deaths have been reported to the Secretary of the Institute during the year 1908:

Date of Election.	Name.	Date of Decease.
1895.	*Ai Arthur Abbott,	August 25, 1908.
1905.	*Peter T. Austen,	December 30, 1907.
1897.	**William Beals, Jr.,	——— —. —.
1906.	*Louis W. Bond,	July 27, 1908.
1881.	*Edward L. Brown,	June 15, 1908.
1890.	*Edward D. Chester,	November 29, 1907.
1906.	*William T. Climo,	October 19, 1907.
1887.	*L. U. Colbath,	May 26, 1908.
1900.	*John T. Conner,	March 17, 1908.
1901.	†Sterling B. Cox,	May 22, 1908.
1891.	**George Davey,	December 20, 1907.
1879.	*Pat Doyle,	March 27, 1907.
1902.	*Adolf Ekman,	March 14, 1907.
1885.	*Francis T. Freeland,	January 26, 1908.
1896.	*Robert J. Forsythe,	May 23, 1907.
1900.	*James B. Gallagher,	January —, 1908.
1898.	*Oliver S. Garretson,	March 18, 1908.
1889.	*T. R. Gue,	July —, 1907.
1903.	**James D. Hague,	August 3, 1908.
1897.	*Frederick S. Harris,	July 17, 1908.
1904.	*Max Heberlein,	November 3, 1908.
1889.	*Albert Helms,	May 28, 1907.
1898.	*Alfred E. Jessup,	November —, 1907.
1902.	*Ralph L. Johnson,	July 8, 1908.
1906.	*Tom Cobb King,	February 27, 1908.
1905.	*Clermont Livingston,	October 20, 1907.
1884.	*Jawood Lukens,	March 10, 1908.
1906.	*Hoyt S. McComb,	February 26, 1907.
1903.	*John McConnell,	October 22, 1906.
1905.	**Donald R. Morgan,	July 25, 1908.
1906.	*E. G. N. North,	March 29, 1908.
1907.	*M. N. Srinivas Rao,	April 13, 1908.
1903.	*Francis A. Roepper,	——— —, —.
1907.	*Edward F. Schaefer,	May 27, 1908.
1901.	**Herbert P. Seale,	September 21, 1908.
1905.	*William Seaton, Jr.,	January 18, 1908.
1905.	*Fred J. Shaler,	March 16, 1908.
1901.	*Harry L. Shrom,	January 18, 1908.
1902.	*Egbert Smit,	June 26, 1908.
1907.	*Charles J. Steffens,	September 18, 1908.
1901.	*Henry Stern,	November 5, 1908.
1900.	**Ernst A. Thies,	August 10, 1908.
1902.	*Herbert N. Tod,	October 30, 1908.
1890.	*Ralph W. Watson,	August 26, 1907.
Hon.	*Prof. Dr. Hermann Wedding,	May 6, 1908.
1890.	*Cabell Whitehead,	September 7, 1908.
1883.	*John Wilkes,	July 6, 1908.
1900.	*Cary Wright,	August 26, 1908.
1895.	*William S. Yeates,	February 18, 1908.

* Member. ** Life Member. † Associate.

Proceedings of the Ninety-Fourth Meeting, New York, February, 1908.

THIS meeting was held at the home of the Institute in the Engineering Societies Building, 29 West 39th St., New York, N. Y., Feb. 18 to 21, 1908.

The first session, held in the large auditorium Tuesday evening, Feb. 18, was called to order at 8 p.m. by Prof. Henry M. Howe, past-President of the Institute, who welcomed the members and guests in a few cordial and well-chosen remarks.

The following papers, accompanied by lantern-illustrations, were presented in oral abstract by their authors:

Luther, Körner, Humboldt, and Swedenborg, by R. W. Raymond, New York, N. Y.

Electric Power in Steel-Mills, by David B. Rushmore and Karl A. Pauly, Syracuse, N. Y.*

At the close of the session, an informal reception in the offices of the Institute was tendered to members and guests.

The second session, held in Assembly Room No. 1 in the same building, Wednesday, Feb. 19, was called to order at 10 a.m. by Prof. Robert H. Richards, past-President of the Institute. Before proceeding with the papers on iron and steel, which were scheduled for presentation at this session, F. E. Junge, a representative of the gas-engine and electric-motor industry of Berlin, Germany, gave a short oral discussion of the paper by Mr. Rushmore and Mr. Pauly, Electric Power in Steel-Mills.*

Printed copies of the paper, The Uniform Nomenclature of Iron and Steel,† were distributed among the members and guests present, and the request was made for a later discussion of this important subject by letter. This paper constitutes the Report of Committee No. 24, of the International Association for Testing Materials, which was presented at the Brussels Congress in 1906.

* Manuscript not received in time for publication in this volume.

† *Bi-Monthly Bulletin*, No. 20, March, 1908, pp. 227 to 237. Not included in this volume.

The following paper was presented in oral abstract by the author:

The Work of the Testing Department of the Watertown Arsenal, in Its Relation to the Metallurgy of Steel, by James E. Howard, Watertown, Mass.

An animated discussion of Mr. Howard's paper then followed, part of which was by letter and part in person. In the absence of the authors, the Secretary presented an oral abstract of the contributions from R. W. Mahon, of the New York Central & Hudson River Railroad Co., West Albany, N.Y.; J. A. Kinkead, of the American Locomotive Co., Schenectady, N. Y.; Charles L. Huston, of the Lukens Iron & Steel Foundry, Coatesville, Pa.; F. N. Speller, of the National Tube Co., Pittsburg, Pa.: and Charles S. Churchill, of the Norfolk & Western Railway, Roanoke, Va. J. P. Snow, of the Boston & Maine Railroad, Boston, Mass., discussed the paper orally, also Mr. Howard and A. A. Stevenson, Burnham, Pa.

In the absence of the author, the Secretary presented an oral abstract of a discussion by Dr. P. H. Dudley, New York, N. Y., of Professor Howe's paper, Piping and Segregation in Steel Ingots, Preliminary Paper,* which was followed by an oral discussion by Professor Howe, New York, N.Y.; Hiram W. Hixon, Philadelphia, Pa.; A. A. Stevenson, Burnham, Pa.; and Prof. William Campbell, New York, N. Y.

The third session was called to order at 2.30 p.m. by Dr. James Douglas, past-President of the Institute.

The following paper was presented in oral abstract by its author:

Present Mining Conditions on the Rand, by Thomas H. Leggett, New York, N. Y. Mr. Leggett's paper was briefly discussed by Alfred James, London, England, President of the Institution of Mining and Metallurgy.

The following paper was presented in oral abstract by the Secretary in the absence of the author:

The Chinese on the Rand, by T. Lane Carter, Johannesburg, South Africa.

The following paper was presented in oral abstract by the author:

* *Trans.*, xxxviii., 3 to 108; 924 to 935.

The Coal-Briquette Plant at Bankhead, Alberta, Canada, by Edward W. Parker, Washington, D. C. Mr. Parker's paper was discussed orally by William H. Blauvelt, Syracuse, N. Y.; Dr. James Douglas, New York, N. Y.; C. G. Atwater, New York, N. Y.; and Mr. Parker.

Then followed a continued discussion of Professor Howe's paper, Piping and Segregation in Steel Ingots, by Professor Howe and A. A. Stevenson.

The following paper, illustrated by a working-model, was presented in oral abstract by the author:

Compression of Semi-Liquid Steel Ingots, by N. Lilienberg, Philadelphia, Pa.*

The fourth and concluding session, held in the same place, was called to order at 8 p.m. by Dr. James Douglas, past-President of the Institute.

The following papers, illustrated by lantern-views, were presented in oral abstract by their authors:

The Physical Features and Mining Industry of Peru, by George I. Adams, Washington, D. C.

The Wilfley Table, II., by Robert H. Richards, Boston, Mass.

The following papers were presented in oral abstract by their authors:

The Investigations of Fuels and Structural Materials by the Technologic Branch of the U. S. Geological Survey, by Joseph A. Holmes, Washington, D. C.†

The Cause of the Explosion in the Monongah Coal-Mine, at Monongah, W. Va., by Frank Haas, Monongah, W. Va.‡

A Possible Explanation of the Occurrence of Secondary Mine-Explosions, by W. O. Snelling, Washington, D. C.‡

Sulphur Dioxide as an Agent in Fighting Mine-Fires, by W. O. Snelling, Washington, D. C.

The following papers were presented in printed form:

The Central Power-Station of the De Beers Consolidated Mines, Ltd., Kimberley, South Africa, by Percy A. Robbins, New York, N. Y.

* *Journal of the Franklin Institute*, vol. clxv., No. 2, pp. 121 to 140 (February, 1908). Not included in this volume.

† Published in *Bi-Monthly Bulletin*, No. 22, July, 1908, pp. 531 to 550. Not included in this volume.

‡ Not furnished for publication.

The Bogoslovsk Mining Estate, by William H. Shockley, Tonopah, Nev.

Dip and Pitch, by R. W. Raymond, New York, N. Y.

Genesis of the Lake Valley, New Mexico, Silver-Deposits, by Charles R. Keyes, Socorro, N. M.

Calculation of Mine-Values, by R. B. Brinsmade, Platteville, Wis.

Discussion of the paper of G. B. Lee, The Corrosion of Water-Jackets of Copper Blast-Furnaces, by William Kent, Syracuse, N. Y.; James Douglas, New York, N. Y.; Hiram W. Hixon, Victoria Mines, Ont., Can.; and George D. Van Arsdale, New York, N. Y.*

Diamonds in Arkansas, by Dr. George F. Kunz, New York, N. Y., and H. S. Washington, Locust, N. J.

Blast-Furnace Practice (Supplementary Note), by T. F. Witherbee, Durango, Mexico.†

Biographical Notice of Thomas Septimus Austin, by Arthur S. Dwight, New York, N. Y.‡

The following papers were presented in manuscript form:

Charcoal and Coke as Blast-Furnace Fuels, by R. H. Sweetser, Columbus, Ohio.

Primary Gold in a Colorado Granite, by John B. Hastings, Denver, Colo.

Origin of Pegmatite, by John B. Hastings, Denver, Colo.

Volcanic Waters, by John B. Hastings, Denver, Colo.

The Mechanical Preparation of Ores in Sardinia, by Erminio Ferraris, Monteponi, Sardinia, Italy.

Discussion of the paper of A. L. Sweetser, Chlorination of Gold-Ores; Laboratory-Tests,§ by Charles H. White, Cambridge, Mass.

The following papers were read by title for future publication:

The Mineral Resources of Korea, by Hallet R. Robbins, London, England.

Discussion of the paper of R. G. Brown, The Vein-System of the Standard Mine, Bodie, Cal.,‖ by H. W. Turner, Portland, Ore.

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* *Trans.*, xxxviii., 877 to 884.

† *Trans.*, xxxviii., 900, 901.

‡ *Trans.*, xxxviii., 406 to 411.

§ *Trans.*, xxxviii., 236 to 244.

‖ *Trans.*, xxxviii, 343 to 357.

The Future Gold-Output of Colombia, by Henry G. Granger, Cartagena, Colombia.

The Carbon-Iron Diagram, by Henry M. Howe, New York, N. Y.

Excursions and Entertainments.

An account of the excursions and entertainments in which members and guests of the Institute participated was published in *Bi-Monthly Bulletin*, No. 20, March, 1908, pp. 243 to 247.

At this meeting there were distributed, through the courtesy of the American Society of Mechanical Engineers, copies of an illustrated pamphlet entitled, The Tunnels of the Hudson Companies, by S. D. V. Burr, New York, N. Y.* Also, an illustrated pamphlet entitled, The Manhattan Cross-Town Tunnels of the Pennsylvania Railroad.† These two reprints, together with the paper of H. T. Hildage, New York, N. Y., entitled, Mining Operations in New York City,‡ and the supplementary paper entitled, New York Tunnel Extension of the Pennsylvania Railroad System, by William Couper, New York, N. Y.,§ give a comprehensive idea of the tunnel-work, from both a popular and a technical point of view. These papers are not included in this volume.

List of Members and Guests Registered at Headquarters, or Attending the Excursions.

(Doubtless incomplete.)

Acker, E. O'C., South Bethlehem, Pa.
Adams, George I., Washington, D. C.
Amsden, F. F., Harrisburg, Pa.
Atwater, C. G., New York, N. Y.
Austin, A. M., New York, N. Y.
Barron, G. D., New York, N. Y.
Barron, Mrs. G. D., New York, N. Y.
Birkinbine, John, Philadelphia, Pa.
Blauvelt, W. H., Syracuse, N. Y.
Blow, J. J., Brooklyn, N. Y.
Böhler, R. F., New York, N. Y.
Campbell, William, New York, N. Y.
Carr, Henry C., Westville, Cal.
Channing, J. Parke, New York, N. Y.
Christianson, Mr., New York, N. Y.
Church, John A., New York, N. Y.
Church, Mrs. John A., New York, N. Y.
Clark, J. M., Kanawha Falls, W. Va.
Clymer, E. T., Philadelphia, Pa.
Colvocoresses, G. M., New York, N. Y.
Crane, Walter R., New York, N. Y.
Cromwell, R. H., Brooklyn, N. Y.
Douglas, James, New York, N. Y.
Drinker, H. S., South Bethlehem, Pa.
DuBois, Howard, Philadelphia, Pa.
Dufourcq, R. G., New York, N. Y.
Dwight, A. S., New York, N. Y.
Dwight, Mrs. A. S., New York, N. Y.

* *Bi-Monthly Bulletin*, No. 20, March, 1908, pp. 254 to 274.
† *Ibid.*, No. 20, March, 1908, pp. 275 to 288.
‡ *Trans.*, xxxviii., 360 to 397.
§ *Bi-Monthly Bulletin*, No. 22, July, 1908, pp. xlv to lii.

Dwight, Edmund W., Philadelphia, Pa.
Dwight, Theodore, New York, N. Y.
Eilers, Anton, Brooklyn, N. Y.
Eilers, Mrs. Anton, Brooklyn, N. Y.
Eilers, Miss E., Brooklyn, N. Y.
Eilers, Miss M., Brooklyn, N. Y.
Eustis, A. H., Boston, Mass.
Eustis, F. A., Boston, Mass.
Eustis, W. E. C., Boston, Mass.
Foote, Francis S., Jr., Montclair, N. J.
Foster, O. R., Brooklyn, N. Y.
Francis, Lewis W., New York, N. Y.
Frazer, Persifor, Philadelphia, Pa.
Freeman, W. W., New York, N. Y.
Freeman, Miss, New York, N. Y.
Godley, George McM., New York, N.Y.
Gouyard, Gustave M., New York, N. Y.
Haas, Frank, Fairmont, W. Va.
Hamner, Alfred E., Branford, Conn.
Hammer, V. T., Branford, Conn.
Hibbard, Henry D., New York, N. Y.
Hill, R. T., New York, N. Y.
Hindshaw, Henry H., New York, N.Y.
Hindshaw, Mrs. H. H., New York, N.Y.
Hixon, H. W., Philadelphia, Pa.
Hofman, H. O., Boston, Mass.
Holbrook, L., New York, N. Y.
Holbrook, Mrs. L., New York, N. Y.
Holden, E. C., New York, N. Y.
Holmes, Joseph A., Washington, D. C.
Holmes, S. A., St. Louis, Mo.
Hovey, E. O., New York, N. Y.
Hovey, Mrs. E. O., New York, N. Y.
Howard, J. E., New York, N. Y.
Howe, Henry M., New York, N. Y.
Howe, Mrs. Henry M., New York, N.Y.
Howells, George A., New York, N. Y.
Humphrey, Chas., New Brighton, N.Y.
Humphrey, Mrs.C., New Brighton, N.Y.
Humphrey, George S., New York, N.Y.
Humphrey, Mrs. G.S., New York, N. Y.
Huntoon, L. D., New Haven, Conn.
Hutchins, J. P., New York, N. Y.
Hutchinson, E. S., Newtown, Pa.
Hutchinson, Mrs. E. S., Newtown, Pa.
Ihlseng, A. O., New York, N. Y.
Ingalls, W. R., New York, N. Y.
Ingalls, Mrs. W. R., New York, N. Y.
James, A. C., New York, N. Y.
James, Alfred, London, England.
Jameson, A. H., Branford, Conn.
Jenkins, G. N., New York, N. Y.
Jeppson, George N., Worcester, Mass.
Jewett, J. H., New York, N. Y.
Junge, F. E., Berlin, Germany.
Kissam, W. A., New York, N. Y.
Kearley, Miss E. F., New York, N. Y.
Kelly, William, Vulcan, Mich.
Kemp, James F., New York, N. Y.
Kemp, Mrs. James F., New York, N.Y.
Kempton, C. W., New York, N. Y.
Kempton, Mrs. C. W., New York, N.Y.
Kimber, Alfred, New York, N. Y.
King, Paul S., New York, N. Y.
Knox, H. H., New York, N. Y.
Kunz, George F., New York, N. Y.
Kunz, Mrs. George F., New York, N.Y.
Kunz, Miss, New York, N. Y.
Lake, E. F., New York, N. Y.
Lane, J. S., Brooklyn, N. Y.
Leggett, Thomas H., New York, N. Y.
Leverich, H., New York, N. Y.
Lihme, I. P., Cleveland, Ohio.
Lilienberg, N., Philadelphia, Pa.
Lockhart, Edwin, New York, N. Y.
Lockhart, Mrs. E., New York, N. Y.
Lohman, R. W., New York, N. Y.
Lucas, A. F., Washington, D. C.
Mackintosh, Mrs. J. B., New York, N.Y.
Maynard, George W., New York, N.Y.
Meeks, Reginald, New York, N. Y.
Meeks, Mrs. Reginald, New York, N.Y.
Meissner, C. A., New York, N. Y.
Miller, Harry H., Guanajuato, Mexico.
Miller, Mrs. H. H., Guanajuato, Mexico.
Miller, Jesse W., Chihuahua, Mexico.
Morgenroth, G. E., New York, N. Y.
Morse, Henry G., New York, N. Y.
Munson, Miss E., New York, N. Y.
Munson, Miss J., New York, N. Y.
Norris, R. V., Wilkes-Barre, Pa.
Munroe, Henry S., New York, N. Y.
Olcott, E. E., New York, N. Y.
Ormrod, George, Emaus, Pa.
Ormrod, Miss, Emaus, Pa.
Orrok, George A., New York, N. Y.
Packard, George A., Boston, Mass.
Parker, Edward W., Washington, D. C.
Pendleton, J. S., Reading, Pa.
Pfordte, Otto F., Rutherford, N. J.
Pickler, Henry, New York, N. Y.
Pitman, S. M., Providence, R. I.
Pitman, Mrs. S. M., Providence, R. I.
Playter, George H., Boston, Mass.

Purman, Misses, New York, N. Y.
Raymond, R. W., New York, N. Y.
Reno, Jesse W., New York, N. Y.
Reno, Mrs. Jesse W., New York, N. Y.
Rice, Calvin W., New York, N. Y.
Richards, Robert H., Boston, Mass.
Robins, Thomas, New York, N. Y.
Robins, Mrs. Thomas, New York, N. Y.
Robinson, C. S., Youngstown, Ohio.
Robinson, Mrs. C. S., Youngstown, O.
Rogers, A. P., New York, N. Y.
Rogers, Allen H., New York, N. Y.
Ruitoba, J. A., New York, N. Y.
Rushmore, David B., Schenectady, N. Y.
Schneider, Albert F., New York, N. Y.
Schneider, Mrs. A. F., New York, N. Y.
Schneider, Miss, New York, N. Y.
Schneider, Gustave, New York, N. Y.
Scott, O. N., Toronto, Canada.
Sharpless, F. F., Westbury, N. Y.
Sharpless, Mrs. F. F., Westbury, N. Y.
Smith, Oberlin, Bridgeton, N. J.
Smith, Miss Winifred, Bridgeton, N. J.
Snelling, Walter O., Washington, D. C.
Speller, F. N., Pittsburg, Pa.
Souder, Harrison, Cornwall, Pa.
Spring, Mr., New York, N. Y.
Stanton, F. McM., Atlantic Mine, Mich.
Stanton, Robert B., New York, N. Y.
Stevenson, A. A., Burnham, Pa.
Stoek, H. H., Scranton, Pa.
Stonestreet, G. D., New York, N. Y.
Storrs, L. S., New Haven, Conn.
Stoughton, Bradley, New York, N. Y.
Struthers, Joseph, New York, N. Y.
Sturgis, Edward B., New York, N. Y.
Tolman, W. H., New York, N. Y.
Tolman, Mrs. W. H., New York, N. Y.
Tonking, William H., New York, N. Y.
Tonking, Mrs. W. H., New York, N. Y.
Torrey, Herbert G., New York, N. Y.
Torrey, Mrs. H. G., New York, N. Y.
Tyrrell, Joseph B., Toronto, Canada.
Uehling, Edward A., Passaic, N. J.
Van Zwaluenberg, A., New York, N. Y.
Walker, Arthur L., New York, N. Y.
Waldo, Leonard, New York, N. Y.
Washington, Henry S., Locust, N. J.
Wenstrom, Olof, Boston, Mass.
Wickware, Francis G., New York, N. Y.
Winchell, Horace V., St. Paul, Minn.

Proceedings of the Ninety-Fifth Meeting, Chattanooga, Tenn., October, 1908.

LOCAL COMMITTEES.

General Reception Committee.—Mr. and Mrs. H. S. Chamberlain, Mr. and Mrs. J. T. Lupton, Mr. and Mrs. T. H. Lasley, Mr. and Mrs. R. S. Faxon, Mr. and Mrs. H. Clay Evans, Mr. and Mrs. C. E. James, Mr. and Mrs. B. F. Thomas, Mr. and Mrs. H. S. Probasco, Mr. and Mrs. W. M. Lasley, Mr. and Mrs. L. W. Llewellyn, Mr. and Mrs. T. G. Montague, Mr. and Mrs. E. B. Thomasson, Mr. and Mrs. C. F. Milburn, Mr. and Mrs. C. A. Raht, Mr. and Mrs. C. C. Nottingham, Mr. and Mrs. Z. C. Patten, Mr. and Mrs. Willard Warner, Mr. and Mrs. S. R. Read, Mr. and Mrs. Carl White, Mr. and Mrs. M. M. Allison, Mr. and Mrs. C. E. Buck, Mr. and Mrs. H. F. Temple, Mr. and Mrs. M. M. Hedges, Mr. and Mrs. X. Wheeler, Mr. and Mrs. John A. Patten, Mr. and Mrs. N. T. Montague, Mr. and Mrs. R. H. Williams, Mr. and Mrs. T. R. Preston, Mr. and Mrs. Morrow Chamberlain, Mr. and Mrs. T. C. Thompson, Mr. and Mrs. T. L. Montague, Mr. and Mrs. O. L. Hurlbut, Mr. and Mrs. C. A. Lyerly, Mr. and Mrs. Garnett Andrews, Mr. and Mrs. M. B. Ochs, Mr. and Mrs. Webster James, Mr. and Mrs. D. P. Montague, Mr. and Mrs. Z. C. Patten, Jr., Mr. and Mrs. W. A. Sadd, Mr. and Mrs. J. B. Pound, Mr. and Mrs. George D. Lancaster, Mr. and Mrs. J. F. Johnston, Mr. and Mrs. R. W. Barr, Mr. and Mrs. W. B. Davis, Mr. and Mrs. Douglas Baxter, Mr. and Mrs. W. R. Crabtree, Mr. and Mrs. Lewis M. Coleman, Mr. and Mrs. Newell Sanders, Mr. and Mrs. E. Y. Chapin, Mr. and Mrs. W. B. Riddell, Mr. and Mrs. J. B. Frazier, Mr. and Mrs. John A. Moon, Mr. and Mrs. C. D. Mitchell, Dr. and Mrs. J. H. Race, Mr. and Mrs. A. W. Chambliss, Mr. and Mrs. J. P. Smartt, Mr. and Mrs. J. F. Tarwater, Mr. and Mrs. L. D. Drewry, Capt. and Mrs. A. B. Putnam, Mrs. Minnie C. Ewing, Mr. G. F. Meehan, Mr. Sam S. Gerstle, Mr. P. A. Brawner, Mr. C. M. Preston, Mr. George M. Guild, Mr. L. M. Mansfield, Dr. J. W. Johnson, Dr. J. M. Hogshead.

Executive Committee.—H. S. Chamberlain, *Chairman;* H. S. Probasco, C. M. Preston, H. Clay Evans, J. T. Lupton, M. B. Ochs, C. C. Nottingham, M. M. Allison, R. S. Faxon, C. E. James, C. A. Raht.

Excursion to Chickamauga Park.—T. C. Thompson, *Chairman;* B. F. Thomas, G. F. Meehan, Morrow Chamberlain, W. J. Bass, J. K. McGoodwin.

Excursion to Hale's Bar Lock and Dam.—R. H. Williams, *Chairman;* John A. Patten, Newell Sanders, George M. Guild, Capt. A. B. Putnam.

Excursion to Rockwood.—F. H. Clymer, *Chairman;* T. L. Brown, T. A. Wright, J. P. Tarwater, T. L. Peterman.

Institute Headquarters.—The Hotel Patten.

The first session, held Thursday evening, Oct. 1, in the Banquet Hall of the Hotel Patten, was called to order by Capt. H. S. Chamberlain, Chairman of the Local Committee, who, in behalf of the city of Chattanooga, extended a hearty welcome to the members and guests of the Institute.

Prof. John Hays Hammond, President of the Council, delivered an address on Professional Ethics.

Dr. R. W. Raymond, Secretary, made appropriate remarks concerning James D. Hague, of New York,* and John Wilkes, of Charlotte, N. C.,† eminent members of the Institute recently removed by death.

The second session was held at the same place on Friday morning, Oct. 2, President Hammond presiding.

The following papers were presented in oral abstract by the authors:

The Treatment of the Gold-Ores of Hog Mountain, Alabama, by T. H. Aldrich, Jr., Birmingham, Ala. This paper was discussed by R. W. Raymond, New York, N. Y.; P. A. Maignen, Philadelphia; and Mr. Aldrich.

Experience with the Gayley Dry Blast at the Warwick Furnaces, Pottstown, Pa., by Edward B. Cook, Pottstown, Pa. This paper was discussed by Edgar S. Cook, and Edward B. Cook,‡ Pottstown, Pa.

Gayley's Invention of the Dry Blast, by R. W. Raymond, New York, N. Y.

The Relation of Slow Driving to Fuel-Economy in Iron Blast-Furnace Practice, by John B. Miles, Philadelphia, Pa.

The third session was held at the same place, Friday evening, Oct. 2, President Hammond presiding.

The following papers, illustrated by lantern-slides, were presented in oral abstract by the authors:

The Geology of the Alabama Coal-Measures, by Charles Butts, Washington, D. C.‡

The Clinton Iron-Ore Deposits in Alabama, by Ernest F. Burchard, Washington, D. C.§

The fourth and concluding session was held in the same place, Saturday morning, Oct. 3, President Hammond presiding.

The following papers were presented in oral abstract by the authors:

* This volume, frontispiece and pp. 677 to 685.

† *Bulletin* No. 25, January, 1909, pp. xxvi to xxvii.

‡ Not furnished for publication.

§ Not included in this volume, but will be published in Volume XL.

Monazite and Monazite-Mining in the Carolinas (presented by Mr. Pratt), by Joseph Hyde Pratt, Chapel Hill, N. C., and Douglas B. Sterrett, Washington, D. C.*

The Clinton Iron-Ore Deposits in Stone Valley, Huntingdon County, Pa., by J. J. Rutledge, Baltimore, Md. This paper was discussed by H. S. Chamberlain, Chattanooga, Tenn., and William Kelly, Vulcan, Mich.*

The Lock and Dam at Hale's Bar, by Major William W. Harts, Corps of Engineers, U. S. Army.†

Studies of Illinois Coals (presented by H. Foster Bain), by H. Foster Bain, Frank W. DeWolf, J. M. Lindgren, Perry Barker, George S. Rice, J. M. Snodgrass, A. Bement, W. F. Wheeler, and C. K. Francis.*

Investigation on Jigging, by Royal P. Jarvis, Knoxville, Tenn.

In addition to the papers already noted, the following were presented in printed form, available for the use of members:

An Unusual Blast-Furnace Product; and Nickel in Some Virginia Iron-Ores, by Frank Firmstone, Easton, Pa.

A New Theory of the Genesis of Brown Hematite-Ores; and a New Source of Sulphur Supply, by H. M. Chance, Philadelphia, Pa.

The Distribution of the Elements in Igneous Rocks, by Henry S. Washington, New York, N. Y.

Requirements of a Breathing-Apparatus for Use in Mines, by Walter E. Mingramm, New York, N. Y.

The Ilsede Hütte Iron-Mines at Peine, Germany, by Lucius W. Mayer, New York, N. Y.

Gold-Dredging on the Choco Rivers, Republic of Colombia, South America, by Henry G. Granger, Cartagena, Colombia.

The Silver-Mines of Mexico, by Albert F. J. Bordeaux, Thonon-les-Bains, Savoie, France.

The Kaffir Mine-Laborer, by T. Lane Carter, Johannesburg, South Africa.

The Mining and Milling of Silver-Lead- and Zinc-Ores at Pierrefitte Mines, France, by William W. Van Ness, London, England.

* Not included in this volume, but will appear in Volume XL.

† *Bi-Monthly Bulletin*, No. 24, November, 1908, pp. 1192 to 1195. Not included in this volume.

The Hardinge Conical Pebble-Mill, by H. W. Hardinge, New York, N. Y.

Sulphur in Gaseous Fuels, by F. Louis Grammer, Leesburg, Va.

The Effect of Humidity on Mine-Explosions, by Carl Scholz, Chicago, Ill.

The following papers were presented in manuscript form:

The Pearce Gold-Separation Process, by Harold V. Pearce, London, England.

The Constitution of Copper-Iron and Copper-Lead-Iron Mattes, by Charles H. Fulton and Ivan E. Goodner, Rapid City, S. D.

The Behavior of Calcium Sulphate at Elevated Temperatures with Some Fluxes, by H. O. Hofman and W. Mostowitsch, Boston, Mass.

The Professional Examination of Undeveloped Mineral Properties, by Charles Catlett, Staunton, Va.

Philippine Coal-Fields, by J. B. Dilworth, Philadelphia, Pa.

The Ozark Lead- and Zinc-Deposits: Their Genesis, Localization, and Migration, by Charles R. Keyes, Des Moines, Iowa.*

A Labor-Chart for the Management of Mining and Milling Operations, by Joseph Macdonald, Guanajuato, Mexico.

Development-Sampling and Ore-Valuation of Gold-Mines, by C. Baring Horwood, Johannesburg, South Africa, and Mungo Park, Glan Ceris, North Wales, Great Britain.

The Air-Furnace Process of Preparing White Cast-Iron for the Malleablizing Process, by Henry M. Howe, New York, N. Y., and Enrique Touceda, Albany, N. Y.

Discussion of the paper of W. F. Wheeler, Pure Coal as a Basis for the Comparison of Bituminous Coals, by A. Bement, Chicago, Ill.

Discussion of the paper of Dr. James Douglas, Secrecy in the Arts, by Edgar Hall, Silverspur, Queensland, Australia.

Discussion of the paper of T. H. Leggett, Present Mining Conditions on the Rand, by W. Fischer Wilkinson, London, England.

Discussion of the paper of R. W. Raymond, Dip and Pitch, by R. W. Raymond; Henry Louis, Newcastle-on-Tyne, England; E. H. Williams, Jr., Woodstock, Vt.; Algernon Del Mar,

* Not included in this volume, but will be published in Volume XL.

Pasadena, Cal.; W. Spencer Hutchinson, Boston, Mass.; C. T. Mixer, Salt Lake City, Utah; Victor C. Hills, Denver, Colo.; and Harold N. Lawrie, White Salmon, Wash.

Discussion (additional) of the paper of George B. Lee, The Corrosion of Water-Jackets of Copper Blast-Furnaces,* by C. D. Demond, Anaconda, Mont.; A. S. Dwight, New York, N.Y.; and J. A. Thomson, Pullman, Wash.

Discussion of the paper of Charles R. Keyes, Genesis of the Lake Valley, New Mexico, Silver-Deposits, by Bernard MacDonald, Guanajuato, Mexico.

Discussion of the paper of W. H. Shockley, The Bogoslovsk Mining Estate, by H. W. Mussen, Collingwood, Ont., Canada.

Discussion (additional) of the paper of Henry M. Howe, Piping and Segregation in Steel Ingots,† by Henry D. Hibbard, Plainfield, N. J., and Mr. Howe.

Discussion of the report, Uniform Nomenclature of Iron and Steel,‡ by Henry D. Hibbard, Plainfield, N. J.; William Kent, Syracuse, N. Y.; and Henry M. Howe, New York, N. Y.

Discussion (additional) of the paper of Mr. Howard, The Work of the Testing Department of the Watertown Arsenal, in Its Relation to the Metallurgy of Steel, by E. H. McHenry, New Haven, Conn.; P. H. Dudley, New York, N. Y.; Henry D. Hibbard, Plainfield, N. J., and Mr. Howard.

The following papers were read by title for future publication:

The Clinton Iron-Ore Deposits in New York State, by D. H. Newland, Albany, N. Y.§

Biographical Notice of James Duncan Hague, by R. W. Raymond, New York, N. Y.

Entertainments and Excursions.

An account of the excursions and entertainments in which the members and guests of the Institute participated was published in *Bi-Monthly Bulletin*, No. 24, November, 1908, pp. 1191 to 1213. This description includes the illustrated pamphlet entitled, Mines and Works of the Tennessee Copper Co., prepared by B. Britton Gottsberger, Copperhill, Tenn., and distributed to those making the trip to Ducktown and Copperhill, Tenn.

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* *Trans.*, xxxviii., 877 to 884.

† *Trans.*, xxxviii., 3 to 108; 924 to 935.

‡ *Bi-Monthly Bulletin*, No. 20, March, 1908, pp. 227 to 237.

§ Not included in this volume, but will be published in Volume XL.

List of Members and Guests Registered at Chattanooga or Attending the Chattanooga Excursions or the Train-Trip to Copperhill and Ducktown.

(Probably incomplete.)

Aldrich, T. H., Jr., Birmingham, Ala.
Allison, M. M., Chattanooga, Tenn.
Allison, Mrs. M. M., Chattanooga, Tenn.
Andrews, Garnett, Chattanooga, Tenn.
Andrews, Mrs. G., Chattanooga, Tenn.
Bailey, Charles, Pittsburg, Pa.
Bain, H. Foster, Champaign, Ill.
Barr, R. W., Chattanooga, Tenn.
Barr, Mrs. R. W., Chattanooga, Tenn.
Bartlett, J. H., New Market, Tenn.
Baxter, Douglas, Chattanooga, Tenn.
Baxter, Mrs. D., Chattanooga, Tenn.
Bone, A. J., Copperhill, Tenn.
Bone, Mrs. A. J., Copperhill, Tenn.
Bowden, E., Birmingham, Ala.
Brawner, P. A., Chattanooga, Tenn.
Brown, T. L., Rockwood, Tenn.
Brown, W. M., Asheville, N. C.
Buck, C. E., Chattanooga, Tenn.
Buck, Mrs. C. E., Chattanooga, Tenn.
Buck, Stuart M., Bramwell, W. Va.
Buell, L. T., Copperhill, Tenn.
Burchard, E. F., Washington, D. C.
Butts, Charles, Washington, D. C.
Caine, Milton A., Copperhill, Tenn.
Chamberlain, H. S., Chattanooga, Tenn.
Chamberlain, Mrs. H. S., Chattanooga.
Chamberlain, Morrow, Chattanooga.
Chamberlain, Mrs. M., Chattanooga.
Chambliss, A. W., Chattanooga, Tenn.
Chambliss, Mrs. A. W., Chattanooga.
Channing, J. Parke, New York, N. Y.
Chapin, E. Y., Chattanooga, Tenn.
Chapin, Mrs. E. Y., Chattanooga, Tenn.
Clack, J. M., Rockwood, Tenn.
Clack, Mrs. J. M., Rockwood, Tenn.
Clymer, F. H., Rockwood, Tenn.
Coffman, Colonel, Rockwood, Tenn.
Coleman, L. M., Chattanooga, Tenn.
Coleman, Mrs. L. M., Chattanooga.
Cook, Edward B., Pottstown, Pa.
Cook, Edgar S., Pottstown, Pa.
Cornell, R. T., New York, N. Y.
Crabtree, W. R., Chattanooga, Tenn.
Crabtree, Mrs. W. R., Chattanooga.
Crewe, L. C., Dayton, Tenn.
Crockard, F. H., Birmingham, Ala.
Davis, W. B., Chattanooga, Tenn.
Davis, Mrs. W. B., Chattanooga, Tenn.
Day, David T., Washington, D. C.
Devereux, J. H., New York, N. Y.
Drewry, L. D., Chattanooga, Tenn.
Drewry, Mrs. L. D., Chattanooga, Tenn.
Edwards, J., Copperhill, Tenn.
Edwards, Mrs. J., Copperhill, Tenn.
Edwards, Miss, Copperhill, Tenn.
Edwards, R. H., Baltimore, Md.
Ellis, E. E., Birmingham, Ala.
Ervin, J. A., Rockwood, Tenn.
Ervin, Mrs. J. A., Rockwood, Tenn.
Evans, H. Clay, Chattanooga, Tenn.
Evans, Mrs. H. Clay, Chattanooga, Tenn.
Evans, H. K., Rockwood, Tenn.
Evans, Mrs. H. K., Rockwood, Tenn.
Ewing, Mrs. Minnie C., Chattanooga,
Fackenthal, B. F., Jr., Easton, Pa.
Fackenthal, Mrs. B. F., Jr., Easton, Pa.
Fairlee, A. M., Copperhill, Tenn.
Faxon, R. S., Chattanooga, Tenn.
Faxon, Mrs. R. S., Chattanooga, Tenn.
Frazier, J. B., Chattanooga, Tenn.
Frazier, Mrs. J. B., Chattanooga, Tenn.
Freeland, W. H., Copperhill, Tenn.
Freeland, Mrs. W. H., Copperhill, Tenn.
Garvin, H. M., Rock Run, Ala.
Gerstle, Sam S., Chattanooga, Tenn.
Gilchrist, J. D., Denver, Colo.
Gordon, C. H., Knoxville, Tenn.
Gottsberger, B. B., Copperhill, Tenn.
Gottsberger, Mrs. B. B., Copperhill.
Grasty, J. S., Charlottesville, Va
Guess, G. A., Copperhill, Tenn.
Guess, Mrs. G. A., Copperhill, Tenn.
Guild, George M., Chattanooga, Tenn.
Haggard, F. G., Rockwood, Tenn.
Haggard, Mrs. F. G., Rockwood, Tenn.
Harts, Maj. Wm. W., Chattanooga, Tenn.
Hayes, C. Willard, Washington, D. C.
Hammond, John Hays, New York, N. Y.
Hedges, M. M., Chattanooga, Tenn.
Hedges, Mrs. M. M., Chattanooga, Tenn.
Heinz, Mr., Copperhill, Tenn.

Higgins, Edwin, New York, N. Y.
Hogshead, J. M., Chattanooga, Tenn.
Holbrook, L., New York, N. Y.
Holbrook, Mrs. L., New York, N. Y.
Hurlbut, O. L., Chattanooga, Tenn.
Hurlbut, Mrs. O. L., Chattanooga, Tenn.
Hutchinson, E. S., Newtown, Pa.
Hutchinson, Mrs. E. S., Newtown, Pa.
Irving, J. D., New Haven, Conn.
James, C. E., Chattanooga, Tenn.
James, Mrs. C. E., Chattanooga, Tenn.
James, Webster T., Chattanooga, Tenn.
James, Mrs. W. T., Chattanooga, Tenn.
Jarvis, R. P., Knoxville, Tenn.
Johnson, J. E., Jr., Glen Wilton, Va.
Johnson, J. W., Chattanooga, Tenn.
Johnston, J. F., Chattanooga, Tenn.
Johnston, Mrs. J. F., Chattanooga.
Jones, E. R., Las Esperanzas, Mexico.
Kelly, W., Vulcan, Mich.
Kelly, Mrs. W., Vulcan, Mich.
Kirchhoff, Charles, New York, N. Y.
La Follette, H. M., La Follette, Tenn.
Lamoreaux, W. F., Isabella, Tenn.
Lamoreaux, Mrs. W. F., Isabella, Tenn.
Lancaster, G. D., Chattanooga, Tenn.
Lancaster, Mrs. G. D., Chattanooga.
Lasley, T. H., Chattanooga, Tenn.
Lasley, Mrs. T. H., Chattanooga, Tenn.
Lasley, W. M., Chattanooga, Tenn.
Lasley, Mrs. W. M., Chattanooga, Tenn.
Leeper, Mrs. B., Rockwood, Tenn.
Lindgren, W., Washington, D. C.
Llewellyn, L. W., Chattanooga, Tenn.
Llewellyn, Mrs. L. W., Chattanooga.
Lodge, J., South Pittsburg, Tenn.
Lupton, J. T., Chattanooga, Tenn.
Lupton, Mrs. J. T., Chattanooga, Tenn.
Lyerly, C. A., Chattanooga, Tenn.
Lyerly, Mrs. C. A., Chattanooga, Tenn.
McCallie, S. W., Palmetto, Ga.
Maignen, P. A., Philadelphia, Pa.
Mansfield, L. M., Chattanooga, Tenn.
Martin, R. L., Jr., Pittsburg, Pa.
Meehan, G. F., Chattanooga, Tenn.
Milburn, C. F., Chattanooga, Tenn.
Milburn, Mrs. C. F., Chattanooga.
Miles, J. B., Philadelphia, Pa.
Mitchell, C. D., Chattanooga, Tenn.
Mitchell, Mrs. C. D., Chattanooga.
Montague, D. P., Chattanooga, Tenn.
Montague, Mrs. D. P., Chattanooga.
Montague, N. T., Chattanooga, Tenn.
Montague, Mrs. N. T., Chattanooga.
Montague, T. G., Chattanooga, Tenn.
Montague, Mrs. T. G., Chattanooga.
Montague, T. L., Chattanooga, Tenn.
Montague, Mrs. T. L., Chattanooga.
Moon, John A., Chattanooga, Tenn.
Moon, Mrs. J. A., Chattanooga, Tenn.
Moss, Milton, Huntsville, Ala.
Moubray, J. M., Lewes, England.
Newland, D. H., Albany, N. Y.
Nottingham, C. C., Chattanooga, Tenn.
Nottingham, Mrs. C. C., Chattanooga.
Ochs, M. B., Chattanooga, Tenn.
Ochs, Mrs. M. B., Chattanooga, Tenn.
Parker, E. W., Washington, D. C.
Patten, John A., Chattanooga, Tenn.
Patten, Mrs. John A., Chattanooga.
Patten, Z. C., Chattanooga, Tenn.
Patten, Mrs. Z. C., Chattanooga, Tenn.
Patten, Z. C., Jr., Chattanooga, Tenn.
Patten, Mrs. Z. C., Jr., Chattanooga.
Patton, J. H., Rockwood, Tenn.
Patton, Mrs. J. H., Rockwood, Tenn.
Perry, Robert S., Germantown, Pa.
Peterman, T. L., Rockwood, Tenn.
Peters, Richard, Jr., Birmingham, Ala.
Pound, J. B., Chattanooga, Tenn.
Pound, Mrs. J. B., Chattanooga, Tenn.
Pratt, Joseph Hyde, Chapel Hill, N. C.
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Preston, T. R., Chattanooga, Tenn.
Preston, Mrs. T. R., Chattanooga, Tenn.
Probasco, Mr., Chattanooga, Tenn.
Probasco, Mrs., Chattanooga, Tenn.
Putnam, A. B., Chattanooga, Tenn.
Putnum, Mrs. A. B., Chattanooga.
Race, J. H., Chattanooga, Tenn.
Race, Mrs. J. H., Chattanooga, Tenn.
Raht, C. A., Chattanooga, Tenn.
Raht, Mrs. C. A., Chattanooga, Tenn.
Ramsay, Erskine, Birmingham, Ala.
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Raymond, Mrs. R. W., New York, N. Y.
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Riddell, W. B., Chattanooga, Tenn.
Riddell, Mrs. W. B., Chattanooga.
Rutledge, J. J., Baltimore, Md.
Sadd, J. B., Chattanooga, Tenn.
Sadd, Mrs. J. B., Chattanooga, Tenn.

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Sanders, Mrs. Newell, Chattanooga.
Saunders, W. L., New York, N. Y.
Scaife, H. L., Clinton, S. C.
Sherrerd, John M., Easton, Pa.
Shropshire, W. B., Pittsburg, Pa.
Smartt, J. P., Chattanooga, Tenn.
Smartt, Mrs. J. P., Chattanooga, Tenn.
Stoek, H. H., Scranton, Pa.
Struthers, Joseph, New York, N. Y.
Sylvester, G. E., Rockwood, Tenn.
Sylvester, Mrs. G. E., Rockwood, Tenn.
Tarwater, J. F., Rockwood, Tenn.
Tarwater, Mrs. J. F., Rockwood, Tenn.
Tarwater, P., Rockwood, Tenn.
Temple, H. F., Chattanooga, Tenn.
Temple, Mrs. H. F., Chattanooga.
Thomas, B. F., Chattanooga, Tenn.
Thomas, Mrs. B. F., Chattanooga, Tenn.
Thomasson, E. B., Chattanooga, Tenn.
Thomasson, Mrs. E. B., Chattanooga.
Thompson, T. C., Chattanooga, Tenn.
Thompson, Mrs. T. C., Chattanooga.
Tyssowski, J., Washington, D. C.
Ubbelohde, Dr., Karlsruhe, Germany.
Vaughan, A. E., New York, N. Y.
Warner, Willard, Chattanooga, Tenn.
Warner, Mrs. W., Chattanooga, Tenn.
Wheeler, S. K., Chattanooga, Tenn.
Wheeler, Mrs. S. K., Chattanooga.
White, Carl, Chattanooga, Tenn.
White, Mrs. Carl, Chattanooga, Tenn.
Williams, R. H., Chattanooga, Tenn.
Williams, Mrs. R. H., Chattanooga.
Wilson, J. C., Rockwood, Tenn.
Withenburg, C. W.

PAPERS.

The Carbon-Iron Diagram.

BY HENRY M. HOWE, NEW YORK, N. Y.*

(New York Meeting, February, 1908.)

Part I.

§ 1. *Introduction.* After giving certain definitions, this paper gives the reasons which led to Roozeboom's form of the diagram of the freezing-point curves and general equilibrium of the iron-carbon compounds, steel and cast-iron; and then those which led to replacing it with the present or double diagram, in which graphite is at all temperatures held to be the more stable and cementite the less stable (metastable) form of carbon. Next in § 10 the evidence supporting this greater stability of graphite is examined, then in § 17 that which seems to oppose it, and in § 22 a summary of the evidence is presented.

The second part of the paper considers the topography of the graphite-iron diagram, rejecting summarily the solubility-lines based on the data of Charpy and Grenet and of Mannesmann, on the ground that the data even on their face do not really point toward these lines, and that they are wholly incompetent to determine any solubility-lines whatsoever, because they report together with the dissolved carbon also an indeterminate quantity of cement-carbon, and because even such carbon as is actually dissolved, only in part represents the solubility of graphite, and in another and indeterminate part represents the solubility of cementite, or at least of carbon in presence of cementite as distinguished from graphite.

It might be thought that graphite is but slightly soluble in iron, because of the absence of any suggestions of eutectoid graphite, corresponding to the eutectoid cementite of pearlite in very many cases in which the cast-iron seems to consist of ferrite and massive graphite only, and because of the micrographs of Goerens and Gutowsky. In these the primary austenite of cast-iron quenched but little below the eutectic freezing-

* Professor of Metallurgy in Columbia University in the City of New York.

range, though richly pierced with cementite needles where it adjoins the cementite-austenite eutectic, is free from graphite where it adjoins the graphite-austenite eutectic, indicating that there has been no marked decrease in the solubility of graphite during cooling, and thus suggesting that that solubility itself is very slight.

But on the other hand the very readiness with which the graphite of iron which initially contains little or no cementite changes into cementite on heating and again cooling, certainly indicates that graphite dissolves rapidly, and suggests that its solubility is considerable, because on our present theory that graphite is more stable than cementite, this change can occur only through the dissolving of the graphite and its re-precipitation as cementite. Further evidence is needed to decide this question.

An appendix gives a diagram on Professor Sauveur's plan, representing the relation between the quantity of the different varieties of cementite and the total carbon-content.

§ 2. *Subdivision of the constituents of iron and steel.* The graphite in cast-iron may come into existence at several different stages in the history of the specimen. Kish comes into existence during the early part of the freezing, before the eutectic freezing-point has been reached; some of the graphite is born in the freezing of the eutectic; still another part, such as the temper-graphite of malleable castings, is born at lower temperatures. So with different parts of the cementite and of the ferrite both of steel and of cast-iron. In trying to convey my ideas to others, and in trying to understand theirs, I have been struck with the need of some simple terms which would indicate clearly the genesis of each of these different constituents. Here are certain of these terms.

§ 3. *Definitions. Austenite.* The normal constituent of region 4 of Fig. 1, a solid solution of carbon or of an iron carbide in gamma allotropic iron, is usually spoken of as "gamma iron" or "mixed crystals." When cooled below region 4 it of course tends to split up into ferrite and cementite, but when it contains much carbon, say 1.5 per cent., this decomposition may be restrained in large part by very rapid cooling, so that the solid solution is preserved in the cold but little decomposed. When thus preserved it is called "austenite." It seems to

some, myself included, that our terminology and our explanations would be greatly simplified by extending the meaning of austenite so as to include, not only this, the least decomposed cold form of this solid solution, but also the undecomposed solid solution itself as it exists in region 4 and elsewhere. Therefore I propose the following definition, which I follow in the present paper.

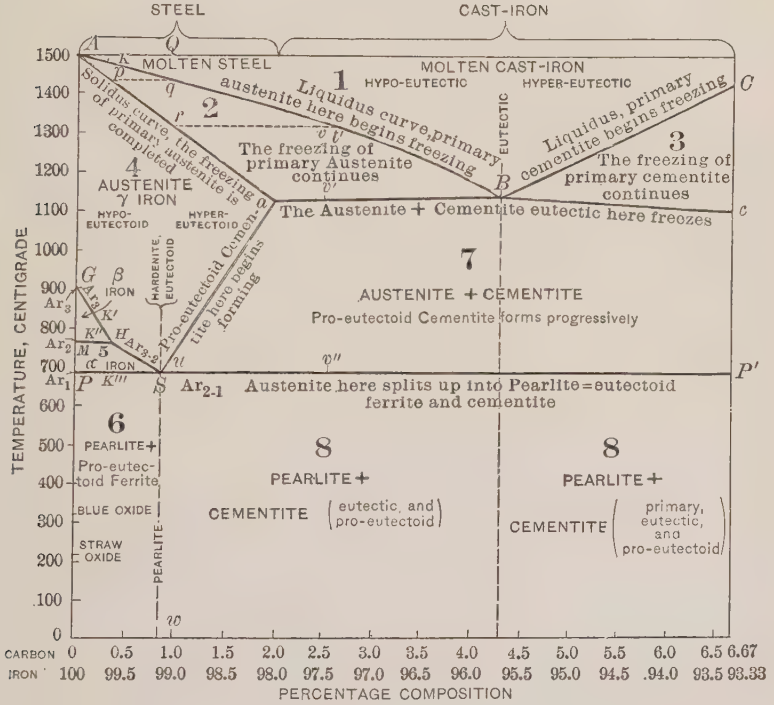

FIG. 1.—ROBERTS-AUSTEN OR CARBON-IRON DIAGRAM. THE CEMENTITE-AUSTENITE OR METASTABLE FORM.

(1) "*Austenite*," (A) *Generically*, the solid solution of iron carbide in iron, stable, for instance, above the transformation-range A_1 to A_3. (B) *Specifically*, this same solid solution as preserved in the cold more or less decomposed, for instance by quenching steel containing more than 1.5 per cent. of carbon from above 1,100° C. in ice-water. When the structure of such steel is developed by polishing on parchment moistened with licorice solution or ammonium nitrate after reheating to 200°, or by etching with hydrochloric acid in an electric current, the austenite remains white, while the zig-zag martensite with which

it is often associated becomes brown. (This paragraph follows approximately the definition of the committee of the Iron and Steel Institute.)[1]

(2) *Primary austenite*, that which separates from the molten metal in cooling through region 2 of Fig. 1.

(3) *Primary cementite*, that which separates from the molten metal in cooling through region 3.

(4) *Eutectic austenite*, that which forms on crossing the boundary *aBc* in cooling from region 2 or 3 into region 7, and forms part of the eutectic.

(5) *Eutectic cementite*, that which forms on crossing the boundary *aBc* in cooling from region 2 or 3 into region 7, and forms part of the eutectic.

(6) *Pro-eutectoid ferrite*, that which forms from the austenite in cooling through region 5. It is the same as "excess ferrite."

(7) *Pro-eutectoid cementite*, that which forms in the austenite in cooling through region 7, and therefore immediately precedes the eutectoid cementite, *i. e.*, that of the pearlite formed on cooling past Ar_1. To the objection that the primary and the eutectic cementite also precede the eutectoid and hence might be called pro-eutectoid, it is a sufficient answer that "ante-bellum" days are understood to include only these shortly before the war in question, and not to include those of Homer, Adam, and the azoic period. There is no reason why it should not be conventionally agreed that "pro-eutectoid" refers only to the stage which precedes the eutectoid immediately.

(8) *Eutectoid ferrite*, that which forms in crossing the line *PSP'*, or Ar_1, in cooling from region 5 or region 7 into region 6 or 8. So with *eutectoid cementite.*

(9) *Primaustenoid*, the network, spines, and other masses rich in ferrite and therefore poor in carbon which, in hypo-eutectoid steel, persist as undiffused relics of the primary austenite formed in cooling through region 2. They are the white masses such as *A*, shown in Fig. 2. As the name suggests, they have the general shape of the austenite, from the decomposition of which in region 5 they result. So long as they persist the heat-treatment or the mechanical work which the metal has undergone cannot be considered as complete, though it may be sufficient for many purposes.

[1] *Journal of the Iron and Steel Institute*, vol. lxi. (1902, No. I.), p. 94.

(10) *Primaustenal*, adj., of or relating to primaustenoid.

Many of these subdivisions must have occurred to other writers; I lay no claim to originality in proposing them. Indeed, some of the names themselves, Nos. 2 to 5, are already in use, either in their present form or in some like form. The only ones which seem to me distinctly new are those numbered 6 to 10 inclusive. My own personal experience as teacher, investigator, writer, and manufacturer convinces me that these subdivisions are useful in discussing and explaining the phenomena.

The pro-eutectoid ferrite is clearly that which Professor Sauveur has called "structurally free ferrite." I have hitherto called it "excess" ferrite. The primary, eutectic, and pro-eutectoid cementite jointly form what Professor Sauveur has called "structurally free" cementite, and I have hitherto called "excess cementite." Indeed, it is the need of subdividing this excess cementite and assigning a specific name, "pro-eutectoid," to that which separates in cooling through region 7, that has led me to propose the name "pro-eutectoid ferrite" instead of "excess ferrite." The former has the merit not only of matching the corresponding cementite, but of indicating the genesis.

The eutectoid ferrite and cementite are clearly those which result from the splitting up of the hardenite and are habitually interstratified as pearlite.

On the same general plan, if the graphite-austenite diagram, Fig. 4 (p. 17), instead of the cementite-austenite diagram is followed, then the graphite which forms in cooling through region III is primary graphite; that which forms on cooling past the line *aBc* is eutectic graphite, etc. Thus "kish" is primary graphite; most of the graphite seen in the fracture of gray cast-iron is probably eutectic; while as to the temper-graphite of malleable cast-iron and of black file-steel something will be said later on in this paper.

§ 4. *Roozeboom on the carbon-iron diagram.* The interpretation of the left-hand part of the diagram, Fig. 1, is relatively simple, because here the only very permanent phases are ferrite, cementite, and austenite. Moreover, there is nothing to suggest that at any time more than two of these are in equilibrium, except at the moment when, in passing across the boundary-line which separates one region from another, one

system is changing into another. Thus in region 4 there is only austenite; in region 5 only austenite and ferrite; in region 6 only ferrite and cementite. With two components, iron and carbon, two phases may be in monovariant equilibrium with each other, *i. e.*, they may form a system which can, without passing out of equilibrium, survive at least a limited rise or fall of temperature; in short, a system which is in equilibrium not simply at some single precise temperature, as is the case with nonvariant systems, but through a range of temperature, though that range may be wide or narrow.

But when the percentage of carbon is so high that graphite forms, a difficulty arises. We still have only two components, iron and carbon, but we have four phases, austenite, graphite, cementite, and ferrite, and in many cases, indeed in most cast-irons, we have three of these phases present simultaneously, austenite, cementite, and graphite above the line PSP' or A_1, and ferrite, cementite, and graphite below the line. But the phase-rule declares that, in a two-component system, not more than two phases can be in monovariant equilibrium.

This conflict with the phase-rule was to be explained away by assuming that what we have to do with here are not really three-phase systems at all, but two different sets of two-phase systems, overlapping through lag, one set with cementite and the other with graphite as a characteristic phase. Two courses were open. Both sets might be equally stable, in which case the stability of each would necessarily be confined to some special range of temperature, because two sets could not be stable simultaneously; or one set might at all temperatures be of less stability than the other, *i. e.*, metastable, and always tending to change over into the other or stable set.

Roozeboom very naturally chose the former theory, because each of these sets certainly does seem at first sight to be stable. Witness the phenomena of two important and familiar processes, first that of cementing steel, and, second, that of "malleablizing" cast-iron, *i. e.*, of changing white and hence cementitiferous cast-iron into gray and hence graphitiferous cast-iron by long heating. In the cementation-process, starting with pure iron and the foreign-graphite which can be used for carburizing it, the two-phase system (?) foreign-graphite + austenite changes into the other two-phase system cementite + austenite,

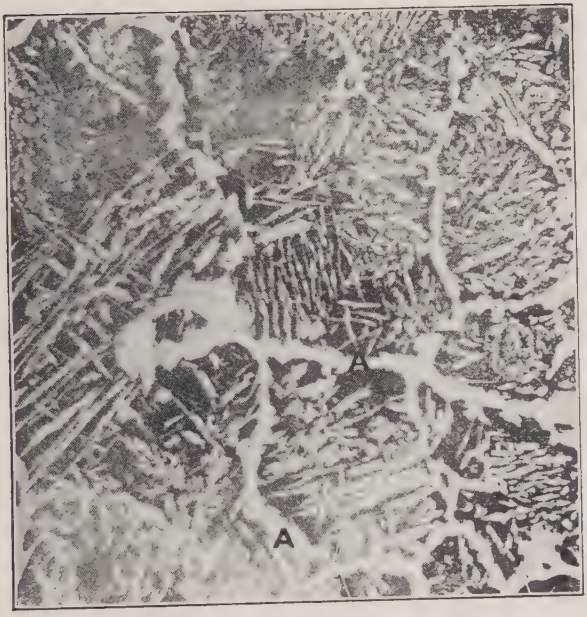

FIG. 2.—PRIMAUSTENOID NETWORK IN UNANNEALED STEEL. A, THE PRIMAUSTENOID. C, 0.42; Si, 0.47; Mn, 0.58 per cent.

FIG. 3A.—No. 3 FOUNDRY-IRON. PROF. WILLIAM CAMPBELL. Silicon, 2; sulphur, 0.05 per cent. 55 diameters. Vertical.

FIG. 5.—THE GRAPHITE-AUSTENITE EUTECTIC. PROF. WILLIAM CAMPBELL.

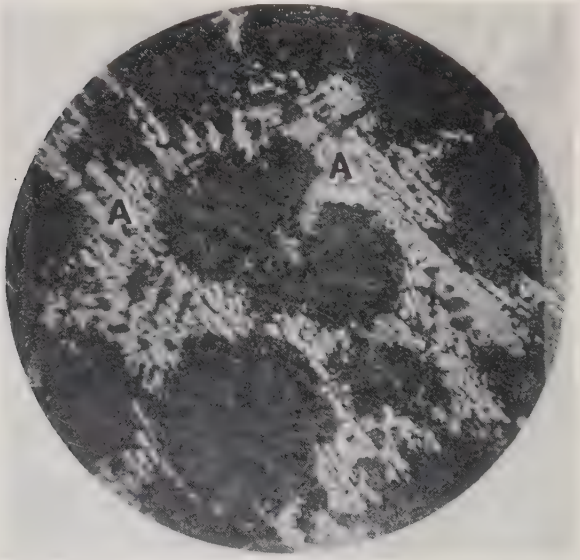

FIG. 6.—CAST-IRON. PROF. WILLIAM CAMPBELL.

The dark circular areas appear to be the graphite-austenite eutectic; the light striped ones, *A*, the cementite-austenite eutectic.

as is implied by the fact that most of the carbon in the cemented steel after cooling usually exists as cementite. Hence, indeed, came the name "cement carbon," and hence in turn the very name "cementite." Here then the system cementite + austenite produces the impression of being not only stable, but more stable than the system foreign-graphite + austenite, because here the latter system actually changes into the former. On the other hand, in the equally familiar process of making malleable cast-iron by holding white cast-iron in region 7, the system cementite + austenite very certainly does change over into the other system graphite + austenite. Here then the latter system is evidently not only stable but more stable than the former. Hence the natural inference that each system in fact is more stable than the other in some certain zone of temperature, and that on crossing the boundary which separates those zones the constitution should change from the system stable in the zone left to the system stable in the zone entered.

What were these two zones? The habitual formation of graphite in the slow solidification of cast-iron certainly suggested that at the freezing-point graphite was a stable phase, and hence that the region in which the system graphite + austenite is stable includes the freezing-point, and hence in turn that the graphite + austenite region lies above the cementite + austenite region. Roberts-Austen had noticed a retardation of the cooling of cast-iron between the freezing-point *aBc* and Ar_1. This retardation ought to represent some reaction or transformation within the solid metal. Putting two and two together, what more natural than to suppose that this represented a change from the system graphite + austenite into the system cementite + austenite by the reaction (1) Gr. + 3Fe = Fe_3C? The phase-rule implies that the boundary-line between two monovariant systems must, in equilibrium, be strictly horizontal, *i.e.*, that the temperature at which one system changes over into the other is a fixed one independent of the proportion between the two phases in either system, for the obvious reason that equilibrium is evidently a thing wholly independent of the relative quantities of the different phases present. This, of course, refers to the temperature at which the transformation is due. That at which it actually occurs may be shifted by lag.

Putting these various things together, we naturally come to

the Roozeboom diagram represented in a general way by Fig. 3. The underscored-V group of freezing-point lines, *ABC*, *aBc*, represented the formation of austenite and graphite from the molten cast-iron; below this line, in region 7A of Fig. 3, the graphite and austenite which had formed in freezing were stable; the retardation observed by Roberts-Austen represented the change from graphite to cementite (1) Gr. + 3Fe = Fe_3C; and in region 7B austenite and this cementite were stable.

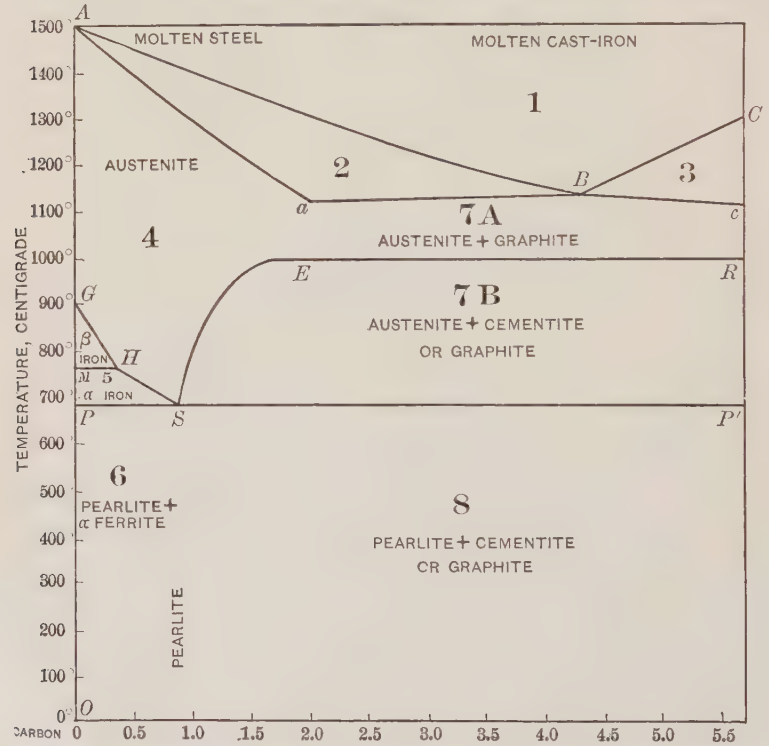

FIG. 3.—GENERAL FEATURES OF ROOZEBOOM'S DIAGRAM OF THE FREEZING-POINT AND EQUILIBRIUM OF THE IRON-CARBON COMPOUNDS.

The graphite habitually found in most cast-irons was simply that which had not had time to change over into cementite, and it was therefore present by lag, and was clearly out of equilibrium. The facts so strongly in contrast that in the cementation-process it is cementite that forms from the union of the iron under treatment and the cementing material, even if this is pure graphite, (1) Gr. + 3Fe = Fe_3C, whereas in the malleablizing process graphite and austenite result from the decom-

position of the cementite originally present (2) $Fe_3C = 3Fe +$ Gr., are explained by supposing that the cementation-process takes place below the line *ER*, and the malleablizing process above that line.

This was a very pretty fabric, and at first sight it seemed to reconcile many conflicting facts. Unfortunately, other evidence soon appeared with which it could not well be reconciled. This Dr. Benedicks has massed together in a very convincing form, adding a wealth of important new observations, which strengthen the present theory immeasurably.[2] Let us now consider some of this evidence, and later let us consider an explanation of the retardation at 1,050°. See § 55, p. 63.

§ 5. *The facts which oppose Roozeboom's theory.* (A) *The reaction* (1) *could not cause a retardation.* A retardation in cooling must of course be caused by some reaction or change which evolves heat, because it necessarily implies that from some source heat is supplied to take the place of part of that which is escaping from the cooling mass. In short, whatever causes this retardation must needs be an exothermic reaction. But the results of Troost and Hautefeuille, later confirmed by E. D. Campbell, show that this reaction, if it took place, would be endothermic. This may further be inferred from Wüst's discovery that the retardation at the eutectic freezing-point, though it bore no constant relation to the carbon-content between the limits of 3.66 and 4.66 per cent., in general increased with the proportion of graphite which formed.[3] His results are rearranged in Table II. so as to show this. It is true that the results of Osmond and Werth indicated that the reaction is exothermic; but their experiments were less conclusive than those of these other observers, and indeed M. Osmond himself accepts the endothermancy of this reaction.

TABLE II.—*The Eutectic Retardation Increases with the Graphite. From Wüst's Data.*

Total carbon-content, per cent....	3.82	3.66	4.04	3.79	3.94	3.76	4.04	4.66
Graphite-content, per cent.....	0.02	0.51	1.75	2.24	2.28	2.33	2.49	3.22
Length of arrest, seconds, at eutectic freezing-point.....	70	120	210	350	290	310	320	270

[2] *Metallurgie*, vol. iii., No. 12, p. 393 (June 22, 1906).

[3] *Metallurgie*, vol. iii., No. 22, p. 757 (Nov. 22, 1906). (I do not know that Prof. Wüst actually draws this inference.—H. M. H.)

§ 6. (B) *The cementation and malleablizing processes do not obey Roozeboom's theory.* Instead of its being true, as this theory implies, that exposure to region 7A of Roozeboom's diagram always leads to the formation of graphite, and exposure to region 7B always to the formation of cementite, the cementation-process normally gives rise to cementite even in region 7A and the malleablizing process of changing cementite into graphite is by no means confined to region 7A, but goes on at much lower temperatures. Thus the normal annealing-temperature in some of the best American malleablizing practice is only 730° (say 1,350° F.), and the process can be carried out even at 677° (say 1,250° F.). Indeed, Charpy and Grenet raised the graphite-content of cast-iron containing 2.10 per cent. of silicon, from 0.10 to 2.83 per cent. by heating it for 6 hr. at 650°,[4] and out of the 2.95 of carbon which Saniter[5] made pure iron absorb by heating in contact with charcoal at about 900°, 0.53 per cent. was graphitic.

§ 7. (C) *The change of combined carbon into graphite takes place at all temperatures between the freezing-point and* 650°, as has now been shown clearly by the concordant and extended experiments of Charpy and Grenet,[4] Heyn and Bauer,[6] and Goerens and Gutowsky,[7] cases 1 to 5, 17 and 18, and 28 of Table I., p. 68, and by other experiments mentioned in § 16.

In cases 1 to 5, when like specimens of cast-iron were held at different temperatures, QT, all the way between 650° and 1,170° and were then quenched so as to retain the carbon in the state there reached, this carbon, which was initially all or nearly all combined, and in large part cementite, was found to have changed in part into graphite at all of these temperatures. This change was in general the more complete the lower the QT, but even at 1,100° and 1,170° it went far.

In cases 17, 18, and 28, when cast-iron was quenched after cooling slowly from above the freezing-point to a series of lower temperatures, QT, the combined carbon changed into graphite progressively as QT was progressively lowered. This graphitization took place all the way from the freezing-point to below

[4] *Bulletin de la Société d'Encouragement pour l'Industrie nationale*, vol. cii., p. 401 (1e semestre, 1902).

[5] *Journal of the Iron and Steel Institute*, vol. lii. (1897, No. II.), p. 122.

[6] *Stahl und Eisen*, vol. xxvii., No. 44, p. 1568 (Oct. 30, 1907).

[7] *Metallurgie*, vol. v., No. 5, p. 145 (Mar. 8, 1908).

900° in case 17, to below 800° in case 18, and to below 750° in case 28.[8] This proof that graphitization is more complete at lower temperatures, 900° and 800°, than at those above 1,000° and 1,100°, directly disproves this theory of Roozeboom's that graphite is stable only above and cementite only below a certain critical temperature near 1,000°. The fact that graphitization was indeed more rapid just below the freezing-point than at lower temperatures is of course readily explained by the greater molecular freedom at the higher temperatures. Thus most of the change from combined carbon to graphite took place in the first 40° below the end of the freezing.

In both the cast-irons which Heyn and Bauer studied, one with between 2.90 and 4.27 per cent. of silicon, and another with between 1.20 and 1.72 per cent. of silicon, but little graphite formed during freezing proper. Both cast-irons froze as white iron, but immediately after freezing was complete graphite formed rapidly. (See § 12 and Table III.)

§ 8. (D) *Microscopic evidence.* If, in the cast-iron which contains both graphite and cementite, that cementite had been formed by the reaction between the graphite, both primary and eutectic, and the austenite present (1), Gr. + $3Fe = Fe_3C$, as Roozeboom's theory implies, and if the graphite present were simply that which had not yet had time thus to become converted into cementite, in short, was simply the residual unconverted core, then manifestly this unconverted core should be surrounded by the cementite resulting from the conversion of the graphite which originally incased that core. But this is

[8] The persistence of graphitization at temperatures below 900° and even 800° is shown more clearly by the arrangement of Heyn and Bauer's results in Table I., which reports the combined carbon left ungraphitized, than in the arrangement shown by the investigators themselves, who report the graphite itself. Graphitization, a change from the combined or cementite and the dissolved carbon into graphite, is limited and regulated not so much by the quantity of graphite present, as by the quantity of carbon which remains dissolved and combined. The graphite actually formed is simply the difference between the total carbon present, which in these cases depends on the conditions above the melting-point and also on how much carbon has been burnt away, and the carbon which remains dissolved and combined. The progress of the shrinkage of volume of a rubber hot-water bag which I am compressing is to be learnt from the volume of water which remains in the bag, from moment to moment, and not from that which is outside the bag, because this includes not only what I have ejected by my compression but also that which ran over while I was filling the bag.

not the case.[9] The graphite plates in such iron may be surrounded by ferrite or by pearlite, but not by cementite. Instead the islands of cementite are at a distance from this graphite. In short, the littoral region about the supposed residual islands of graphite instead of being the part richest in cementite is rather the part poorest in cementite, as if the reaction had been rather a precipitation of graphite out of cementite than a creation of cementite out of graphite, rather (2) $Fe_3C = 3Fe + Gr.$ than (1) $Gr. + 3Fe = Fe_3C$. This is illustrated in Fig. 3A (p. 9), in which the white islands, *A*, of the cementite-austenite eutectic are far separated from the black sheets of graphite.

§ 9. *The present or double diagram theory.* This theory completely rejects the idea of the zones 7A and 7B of the Roozeboom diagram, with graphite the more stable substance in the upper and cementite the more stable in the lower zone, and instead regards graphite as always and under all conditions the more stable and cementite the less stable of the two, or in other words the metastable form, so that cementite is always trying to change into graphite, just as the austenite and martensite of hardened steel are always trying to change into ferrite and cementite. The theory gives us a double diagram, the cementite-austenite diagram which we have seen in Fig. 1, and roughly parallel with it the graphite-austenite diagram shown in Fig. 4, in which certain of the lines of the cementite-austenite diagram are reproduced to facilitate the explanation. This double diagram was, indeed, outlined in Roberts-Austen's[10] early diagram in 1899 at the suggestion of Le Chatelier.

According to this theory, when cast-iron freezes the whole mass may (1) obey the austenite-cementite diagram, so that a cast-iron wholly free from graphite, *i. e.*, a typical white cast-iron results, consisting of ferrite and cementite only, metastable, *i. e.*, relatively unstable; or (2) the whole mass may obey the graphite-austenite diagram so that no cementite forms, and that the

[9] Heyn, *Iron and Steel Magazine*, vol. x., No. 1, p. 50 (July, 1905). Also Benedicks, *Metallurgie*, vol. iii., pp. 472–3 (No. 14, July 22, 1906), and micrograph 25 after p. 432 (No. 13, July 8, 1906).

[10] *Fifth Report to Alloys Research Committee*, read before Institution of Mechanical Engineers, Feb. 9, 1899, Fig. 10, Plate 4, *Proceedings of the Institution of Mechanical Engineers* (1899).

mass as a whole in cooling becomes a mixture of graphite and ferrite, as in ultra or typical gray cast-iron, perfectly stable; or (3), as is usually the case, certain molecules may follow one diagram and certain others may follow the other, so that the mass as a whole consists of two parts, theoretically distinct but in fact usually mixed up, one which has obeyed the cementite-austenite diagram, and hence when cold consists of a mixture of ferrite and cementite, and the other which has obeyed the graphite-austenite diagram, and hence when cold consists of a

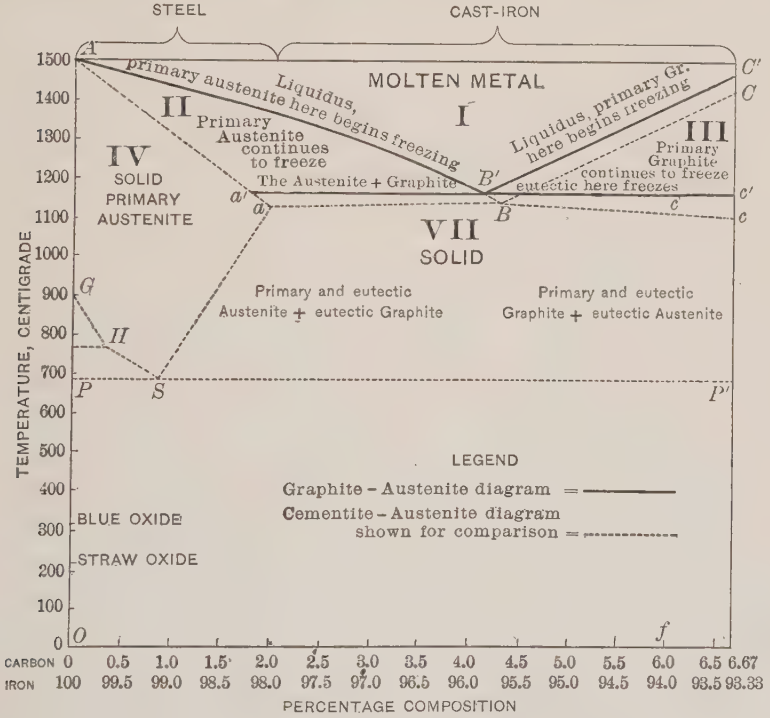

FIG. 4.—GRAPHITE-AUSTENITE OR STABLE CARBON-IRON DIAGRAM.

mixture of graphite and ferrite. Thus the cast-iron as a whole is a conglomerate of these two mixtures. This latter case is well exemplified by mottled cast-iron, with its patches of gray and patches of white.

Let us look briefly first (§ 10) at the facts which evidently support the present theory, that graphite is at all times and under all conditions the more stable and cementite the less stable, so that the presence of graphite in cast-iron is by right and not by lag, and that it is cementite that is present by

lag; let us next (§ 17) consider certain facts which at first seem to oppose this theory.

§ 10. *Evidence of the greater stability of graphite than of cementite.* This theory, of course, accords easily with the phenomena of the malleablizing process, the essence of which is that the mixture of cementite and ferrite, of which white cast-iron initially consists, changes slowly into a mixture of graphite and austenite when long held at a temperature of about 900° C. So when high-carbon steel is held for a long time at a high temperature it undergoes a like graphitization, and loses part of its hardening power.[11] (See K and L, § 16.)

Next come two important facts which argue strongly that graphite is more stable than cementite. These facts are (1) that it is rapid cooling that yields cementite and slow cooling that yields graphite; and (2) that the eutectic cementite, in and immediately after the act of freezing, *i. e.*, very soon after its formation, begins changing into graphite.

Of course it is but natural that the more time is given for reaching stable equilibrium the nearer should be the approach to that equilibrium; hence we naturally infer that graphite, the condition which is reached by slow cooling, is more stable than cementite, that reached by rapid cooling. Hence our first fact argues that graphite is the more stable of the two conditions.

The second fact, the change of the eutectic cementite into graphite, is further evidence that graphite is the more stable of the two, because whatever change occurs is naturally in the direction of the greater stability. Indeed, it is the usual order of things that the metastable form is the one most readily entered, and that this later changes over into the stable form.[12]

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[11] *The Treatment of Steel*, Wm. Metcalf, revised and reprinted by the Crucible Steel Co. of America, p. 88 (1902).

[12] Ostwald says, *Grundriss der allgemeinen Chemie*, p. 312 (1899): "Wenn man, das metastabile Gebiet überschritten hat, und es entsteht eine neue Phase freiwillig, so macht sich das bemerkenswerte Gesetz geltend, *dass die entstehende Form nicht die unter den vorhandenen Umständen beständigste ist, sondern im Gegenteil die wenigst beständige, d. h. die in Bezug auf ihre Beständigkeit der sich umwandelnden Form zunächst liegende. Die Erscheinung ist ausserordentlich verbreitet.*"

An interesting illustration of this law that the form most readily entered is not usually the most stable, has lately been reported by Dr. A. L. Day. We often think

§ 11. *Precedence of the metastable to the stable.* This order is readily understood if we liken it to what happens to a lot of well-loaded arrows which I throw pell-mell out of the window, so that they fall upon the soft asphalt sidewalk. No matter in what position any individual arrow is when it leaves my hand, it will strike and sink into the asphalt head-foremost, *i. e.*, in the vertical or metastable position, and will reach the horizontal or stable position later only when some favoring condition permits the change to occur. For instance, if the weather grows very hot and the asphalt softens, relaxing its frictional resistance, the arrows begin toppling over and approaching the horizontal or stable position, quite as when white cast-iron is highly heated in the malleablizing process the frictional resistance or molecular rigidity is so far relaxed that the metastable cementite is able to obey its natural impulse and topple over into the stable forms of graphite and austenite. Whatever change takes place is in the direction of greater stability. No arrow which has fallen flat will right itself and stand on end, nor should graphite change back into cementite.

In the case of the carbon-iron diagram this order is all the more easily understood if we accept the theory that the carbon in the molten iron exists dissolved as a solution not of elemental carbon, *i. e.*, graphite, but of cementite in the molten iron. The quick freezing does not give time for the change from the cementite to the graphitic state; slow freezing does.

§ 12. *The evidence detailed.* Of the two facts (1) that rapid cooling yields cementite and slow cooling graphite, and (2) that the eutectic cementite early begins changing into graphite, the first is a matter of such common observation that we need not here detail the abundant specific evidence on which it rests. The second is established by the very concordant results of Heyn and Bauer, of Goerens and Gutowsky, and of Osmond, condensed in Table III. (p. 20).

The fact that in No. 18 but very little graphite was present when the slow cooling was interrupted by quenching

that the conditions which exist in rock-forming should give rise to the most stable forms, because here there is such ample time. But the particular form of magnesium silicate, enstatite, which is common in nature, Dr. Day finds is not the stable one. Enstatite and two others of the four forms of this silicate can be changed into the fourth, but this cannot be changed back into any of the others. Therefore this fourth is the stable form. *Fifth Year Book of the Carnegie Institution*, p. 178 (1906).

TABLE III.—*The Formation of Graphite Takes Place Chiefly Just Below the Freezing-Point of the Cementite-Austenite Eutectic.*

No. in Table I.	Authority.	Composition of Iron.					Percentage of Graphite.																
		Total C. Per Cent.	Si. Per Cent.	Mn. Per Cent.	P. Per Cent.	S. Per Cent.	Degrees above the Eutectic Freezing-Point.			Degrees below the Eutectic Freezing-Point, approximately.													
							20°	5°	3°±	0°	.....	11°	18°±	20°±	30° 31°	41°	45°	120°	130°	198°±	220°	370°	510°
17	Heyn and Bauer.[a]	3.05 to 3.33	1.20 to 1.72	0.22±	0.09±	0.33±	.....	.....	0.35	.....	.....	.....	1.36	1.56	.....	.....	.....	.....	.....	2.06	.....	.....	.....
18	Heyn and Bauer.	2.59 to 3.14	2.90 to 4.27	0.22±	0.03±	0.33±	.....	0.36	.....	.....	.....	0.62	.....	.....	1.55	2.06	.....	.....	.....	.....	.....	.....	.....
28	Goerens and Gutowsky.	3.85 to 3.96	0.12	0.18	0.02	0.008	20° above beginning of eutectic freezing. 0 per cent.	.....	at beginning of eutectic freezing. 1.05 1.89	at end of eutectic freezing. 2.61	.....	.....	.....	.....	.....	.....	.....	.....	2.87	.....	.....	.....	.....
	Osmond.[b]	.....	2.13	2.12	0.16	0.04	.....	.....	.....	0.18	.....	.....	.....	.....	.....	.....	2.02	1.96	.....	.....	2.06	2.64	2.65

In all four of these series given in Table III. the cast-iron was cooled slowly from the molten state to the temperature indicated at the top of the several columns, and then quenched in water. In Series 28 the cast-iron, which when quenched at the beginning of the eutectic freezing contained 1.89 per cent. of graphite, was cooled much more slowly than the other, which then had only 1.05 per cent. In this series the temperatures given represent those of the furnace itself rather than of the iron experimented on. This was in such small quantity, only 2 g., that its retardation in freezing could have little effect on the temperature of the thermo-couple, which therefore outran the cooling of the iron. Hence the actual temperature of the iron was above that here given. The assertions that the iron was "At the beginning" or "At the end of the eutectic freezing" are based on direct observation of the physical condition of the iron at the time of quenching.

[a] The temperature which I have here taken as the eutectic freezing-point, 1,105°, is that at or about which the freezing of the eutectic seems to end. It begins at about 1,115°. See Heyn and Bauer, *Stahl und Eisen*, vol. xxvii., No. 44, p. 1566, column 2, lines 6 to 11 (Oct. 30, 1907).

[b] M. Osmond does not give the eutectic freezing-point, which, indeed, was not then recognized. But he noticed a retardation in cooling between 1,085° and 1,130° which very probably represents this point. It is taken here at 1,120° *Études Métallurgiques*, from *Annales des Mines*, vol. xiv., 8th Series, p. 49 (July-August, 1888).

at 11° below the apparent end of the freezing, indicates that the eutectic which actually froze was nearly all cementite-austenite, and in but very small part, if at all, graphite-austenite. The very rapid increase in the graphite-content as the quenching-temperature sinks farther to 18°, 20°, 30°, and 31° below the end of the freezing, is far too great to be explained by the decrease in the solubility of carbon in the austenite, and consequent precipitation of carbon from that austenite, and hence must be due to the change of the eutectic cementite into graphite, Fe_3C = Gr. + austenite, an inference which is confirmed by the micrographs. In these the cementite-austenite eutectic diminishes rapidly in volume as the quenching-temperature is lowered, and is replaced by graphitiferous nests.

The quantitative results in Series 28 of Goerens and Gutowsky are less conclusive, because the excess of graphite at the end over that at the beginning of the eutectic freezing might represent only direct formation of the graphite-austenite eutectic from the molten state. The same objection applies to the evidence of their micrographs, even those which at first seem most favorable to the theory of the formation of graphite out of the eutectic cementite. Thus about half of their cast-iron No. 21, cooled in 7 min. from 1,250° to 1,100° and then quenched, consisted of cementite-austenite eutectic; the other half consisted of primary austenite and graphitic spheres. But their exactly similar No. 22, cooled in 8 min. from 1,250° to 1,080° and then quenched, had none of this cementite eutectic. The place where it had been was now filled with a graphitiferous mass. It seems to me probable that part of the cementite-austenite eutectic found in No. 21 represented metal already solid at the moment before quenching, and hence that its absence in No. 22 implies that in the cooling from 1,100° to 1,080° this cementite had graphitized. But this is only an inference, and not a necessary conclusion, because it is possible to argue that the cementite-austenite eutectic of No. 21 represents solely the metal still molten at the instant before quenching, and that the metal solid at that same instant was free from cementite, so that the freedom of No. 22 from cementite-austenite eutectic means only that the slow cooling from 1,100° to 1,080° allowed the iron, molten at 1,100°, to freeze as graphite-

austenite eutectic instead of freezing as cementite-austenite eutectic, as it did in the quenching of No. 21 at 1,100°.

§ 13. *Can eutectic graphite form directly from the molten state?* The language of Goerens and Gutowsky might be taken to mean that, in their opinion, the graphite which forms in and near the freezing-range forms solely from the decomposition of the eutectic cementite.[13] But of this I find no proof either in their work or elsewhere. Indeed, the structure shown in such a micrograph as Fig. 5 by Professor Campbell (page 10), seems clearly that of a true graphite-austenite eutectic, *i.e.*, one formed direct from the molten state and not from the decomposition of cementite. Goerens and Gutowsky repeatedly speak of finding in their cast-irons a "graphite-eutectic," for instance, an "extraordinarily fine" one in their No. 18; and their No. 22, referred to just above here, consists solely of primary austenite and "partly lamellar graphite-eutectic."[14] It does not seem to me that a pseudomorph, a metamorphic mass formed from the decomposition of the cementite-austenite or any other eutectic, can be spoken of as a graphite-eutectic at all, because this term should be applied solely to a true graphite-eutectic, formed as such direct from the molten state.

Here let interpretation be most cautious. Though much of the graphite probably springs from cementite, some may be part of a true graphite eutectic, as the results of Goerens and Gutowsky themselves suggest. Thus, in describing their experiment 18, in which cast-iron quenched at 1,135°, the beginning of the eutectic freezing-range, contained spheres of "extraordinarily fine graphite eutectic," they point out the extremely important fact, mentioned in § 48, that the primaustenoid pine-trees are crossed with needles of cementite where they adjoin the cementite-austenite eutectic, but are free from these needles where they adjoin the graphite-austenite eutectic. Now, if this latter were derived from the cementite-austenite eutectic, then the pine-trees of primaustenoid which it adjoins should have been pierced by these cementite needles at the time when the assumed cementite-austenite eutectic existed there and before it had changed into this present graphite-austenite eutectic, and these needles should still be present.

[13] *Metallurgie*, vol. v., No. 5 (Mar. 8, 1908), first lines of p. 146.
[14] *Idem*, pp. 141–2.

Evidence of the formation of the graphite-austenite eutectic direct from the molten state is given by Dr. Benedicks,[15] in the shape of spherulites projecting into a vug in cast-iron, and hence evidently formed from the molten state. It is clear that these spherulites consist, at least in large part, of graphite-austenite eutectic, because of their general appearance, which is identical with that of the spheres of this eutectic found by Benedicks and others in cast-iron, and especially because they contain plates of graphite which actually project into the vug, and hence cannot have been formed by the decomposition of cementite.

§ 14. *Another view of the order of solidification.* Prof. William Campbell infers from the structure shown in Fig. 6 (p. 10) that the graphite-austenite masses represented by the dark spheres or circles must have formed earlier than the cementite-austenite eutectic, *A*, shown beside them. This he infers from the fact that the cementite-austenite eutectic surrounds these spheres. But this is hardly conclusive. True, if these spheres form direct from the molten state, their shape suggests their preceding the cementite-austenite eutectic, because it is not a shape likely to be left between the dendrites of an older formation. Yet these spheres may not have formed from the molten state at all, but through the metamorphosis of the earlier formed cementite-austenite eutectic. This may well have been true in case of the graphitic spheres shown in the micrographs of Goerens and Gutowsky, of successive specimens of like cast-iron quenched at different temperatures just below the eutectic freezing-point. As the quenching-temperature was progressively lowered, these graphite-bearing spheres came into existence when it had fallen to below the eutectic freezing-point, and gradually increased in volume until, at the apparent end of the freezing, they formed the whole mass of the steel. (Their experiment 22.) Whatever allowance we may make for the lag of the cooling of the cast-iron behind that of the pyrometer's thermo-couple, it is extremely probable that part at least of the growth of these spheres took place after the end of the freezing, and therefore by feeding on the earlier formed

[15] *Metallurgie*, vol. iii., p. 470–1 (No. 14, July 22, 1906), and Micrograph 17, facing p. 432 (No. 13, July 8, 1906).

cementite-austenite eutectic. But if this was the way in which this part of the growth of these spheres took place, why may it not be the way in which the whole of that growth took place? It is in general unsafe to deduce relative age from the relation of envelope and kernel, because, though the kernel is very often older than its envelope, yet sometimes the envelope is the senior of the kernel, both in metallography and in mineralogy.

But if Professor Campbell's inference should turn out to be true, we could easily understand that, with progressively increasing rigidity, the formation of the graphite-austenite eutectic might come to arrest itself through the pressure which the bulkiness of its graphite sets up, and be succeeded by the formation of cementite-austenite eutectic.

§ 15. *The bulkiness of graphite argues for its stability.* Beyond this it is but natural from an important consideration that graphite should be the more stable. It is a well-established rule that pressure in and by itself favors the formation of the less bulky of two phases.[16] Now the change from the state of cementite into that of the bulky graphite implies much expansion. But this expansion must necessarily set up stress within the solid metal, and thus be resisted by the pressure which it itself sets up. Thus the very fact that this change from cementite into graphite does take place in spite of its being opposed by the pressure which it creates, argues that there is a force at work to overcome this opposition, and that force is a manifestation or symptom of the greater stability of the graphitic than of the cementite state.

It is as when my cane, nearly upright and leaning against the edge of my table, is toppled over by gravity and thrown to the floor in spite of the frictional resistance of the table which tends to prevent its starting; the fact that in spite of this

[16] Jones, *The Elements of Physical Chemistry*, Macmillan, 1903, p. 514. "*Increase in pressure diminishes the volume, and therefore favors the formation of that system which occupies the smaller volume.* Equilibrium is, then, displaced by increase in pressure toward the system which occupies the less volume."

Le Chatelier, *Les Équilibres Chimiques*, p. 210. "*Toute variation d'un facteur de l'équilibre amène une transformation du système qui tend à faire éprouver au facteur considéré une variation de signe contraire à celle qu'on lui a communiquée.*

"C'est-à-dire que toute élévation de température provoque une réaction avec absorption de chaleur, toute élévation de pression une reaction avec diminution de volume, . . ."

frictional resistance it moves from the nearly upright to the horizontal position is proof that the latter is the more stable.

This conception helps us to understand why this mill, if it does grind exceeding small, yet grinds so slowly; and further why it is that the shifting into the graphitic state is more rapid at high temperatures. Is it not this? The very beginning of the formation of the bulky graphite out of the compact cementite sets up pressure, which arrests further formation of graphite until the plastic mass has so far flowed under the pressure as to relieve it, and reduce it to such a mildness that it no longer arrests the further formation of graphite. Then a new lot of carbon forms graphite, and sets up new pressure, which temporarily arrests the graphite-forming, and so forth. At very high temperatures, approaching the melting-point, the metal is so soft and plastic that it yields easily and rapidly to this pressure, with the consequence that the formation of graphite is much less strongly opposed, and hence goes on much faster, than at lower temperatures.

§ 16. *Cases in which cementite changes into graphite.* I will now cite twelve cases (A to L) in which heating changed into graphite part of the carbon which was initially present as cementite. In five of these (A to D and L) the proportion thus changed was very large.

(A) *Bell*[17] increased the graphite-content of white cast-iron from 0.374 to 1.79, or by 1.42 per cent., without important change in the total carbon-content, by a 13-day heating in the hot-blast stove of an iron blast-furnace.

(B) *Royston*,[18] by heating iron, originally containing 3.85 per cent. of combined carbon and no graphite, to 1,030° rapidly and quenching it, transferred 2.35 per cent. of the carbon to the graphitic state, leaving 1.50 per cent. of it in combination.

(C) *Brustlein*,[19] by re-cementing blister-steel which originally contained 1.70 per cent. of carbon, all combined, transferred 1.38 per cent. of this carbon to the graphitic state.

(D) *Forquignon*,[20] by reheating iron containing originally

[17] *Principles of the Manufacture of Iron and Steel*, p. 159 (1884).
[18] *Journal of the Iron and Steel Institute*, vol. li. (1897, No. I.), p. 166.
[19] Osmond, *Contribution a l'Étude des Alliages*, p. 374 (1901).
[20] *Idem*, p. 378.

3.27 per cent. of carbon, all combined, transferred 1.21 per cent. of the carbon to the state of graphite in 72 hr. and 1.69 per cent. in 144 hours.

(E) *Wüst and Schlösser*,[21] on heating white cast-iron, which was initially nearly free from graphite, at different high temperatures for 3 hr. and then apparently cooling slowly, found that the carbon had shifted from the combined to the graphitic state to a degree which increased in general with the temperature and with the silicon-content. Their results are condensed in Table I. With even as little as 0.05 per cent. of silicon a little graphite formed at 1,000°. With 0.13 per cent. of silicon the formation of graphite occurred at 900°. With 2.12 per cent. of silicon, graphite formed at a moderate rate even at 600°, and with 3.15 per cent. it formed rapidly at 600°. A very remarkable result which they reached was that the formation of graphite was in one case practically as rapid at 700° and in another case at 600° as at any higher temperature. In the former case the cast-iron contained 2.12 per cent. of silicon, in the latter 3.15 per cent.

(F) *Arnold* found that, on repeatedly cementing steel, much of the carbon passed to the graphitic state.

(G) In this connection it is of interest to note the results of *Charpy and Grenet*.[22] They found that, once the transfer of carbon from the combined to the graphitic state has started, it goes on at temperatures below that necessary to induce it in a graphiteless iron, quite as a crystal of sodium sulphate immersed in a supersaturated solution of that salt causes rapid solidification, *i. e.*, a rapid passage from the unstable supersaturated liquid state to the stable solid crystallized state, by "nucleus action."

(H) *Charpy* found that though cementation at 650° in potassium cyanide gives rise to cementite without any graphite, yet at higher temperatures the cementite is decomposed and changed into graphite. By cementing in illuminating gas and in carbonic oxide he reached the following results:[23]

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[21] *Stahl und Eisen*, vol. xxiv, No. 19, p. 1120 (Oct. 1, 1904).

[22] *Bulletin de la Société d'Encouragement pour l'Industrie nationale*, vol. cii., p. 401 (1e semestre, 1902).

[23] Sur la Cémentation du Fer, *Comptes rendus*, vol. cxxxvi., No. 17, p. 1000 (Apr. 27, 1903); *Iron and Steel Magazine*, vol. viii., No. 4, p. 301 (Oct., 1904).

TABLE IV.—*Charpy's Experiments in Cementation.*

Material Treated.	Cemented in.	Length of Time.	Temperature.	Carbon Absorbed.	
				Graphite. Per Cent.	Combined Carbon. Per Cent.
No. 1. Steel of 0.09 C., 3 mm. diam.	Illuminating gas.	64 hours.	1,000°.	7.66	0.66
No. 2. Steel filings of 2 per cent. carbon.	Pure carbonic oxide.	2 hours.	900°.	3.27	
No. 3.		4 hours.	900°.	4.90	
No. 4.		1½ hours.	1,000°.	2.66	
No. 5.		36 hours.	1,000°.	8.27	1.00

In later trials, on cementing apparently very pure Swedish wrought-iron in wood-charcoal at 1,000°, he found the following percentage of carbon in the product:[24]

Total Carbon. Per Cent.	Graphite. Per Cent.	Combined Carbon by Difference. Per Cent.
2.50	1.40	1.10
3.10	2.21	0.89

(I) *Saniter*,[25] and also *Mylius*, *Förster*, *and Schöne*,[26] give us further instances of the change from cementite into graphite. Saniter found that cementite, isolated from the iron, formed graphite on heating to 800°, and on melting yielded a graphitiferous metallic button containing 4.27 per cent. of carbon. Mylius, Förster, and Schöne found that this isolated cementite, on melting, formed graphite. But, as Benedicks[27] properly points out, this case is of less weight than the others, because the cementite, in being isolated from the iron, is likely to become partly decomposed, and thus to lose in stability, so that it might then change over into graphite under conditions which would not cause the undecomposed cementite to change thus.

(J) *Tiemann's* results may here be considered. Starting with iron wholly free from silicon and containing 4.271 per cent. of carbon as cementite and 0.255 per cent. as graphite, he failed to transfer any of his carbon from the cementite to

[24] *Revue de Métallurgie*, vol. v., No. 2, Mémoires, p. 78 (Feb., 1908).
[25] *Journal of the Iron and Steel Institute*, vol. lii. (1897, No. II.), p. 115.
[26] *Zeitschrift für anorganische Chemie*, vol. xiii., p. 38 (1897).
[27] *Metallurgie*, vol. iii., No. 13, p. 429 (July 8, 1906).

the graphitic state even on long heating to temperatures between 906° and 1,116°. But when 0.75 per cent. of silicon was present, his graphite, which in this case was initially 0.938 per cent., rose to 1.69 per cent. on heating to 975°, and to 2.795 per cent. on heating to 1,125°.[28]

(K) *Prof. William Campbell* found that graphite formed in steel of 1.61 and of 1.72 per cent. of carbon on heating to 1,070°, in steel of 2.04 per cent. carbon on heating to 1,200°, and in steel of 1.94 per cent. carbon on heating to 950°, in each case followed by slow cooling.[29]

(L) *Arnold and Mc William* found 1.38 per cent. of graphite in steel initially of 1.80 per cent. carbon, after holding it at about 1,000° for about 70 hr. and cooling thence extremely slowly.[30]

It has been a tradition in Sheffield for many years, to my knowledge, that the separation of graphite in high-carbon steel is caused either by too low a finishing-temperature, *i.e.*, by continuing the rolling or hammering till the temperature has sunk unduly low, or by unduly long and high heating for annealing. A possible explanation of the effect of cool rolling is that the strong internal tension (the opposite of pressure), which it causes, may favor the formation of the bulky graphite. Graphite, once started locally thus, might by nucleus action spread throughout much of the cross-section. That cool rolling does cause severe internal tension in pieces of appropriate shape is shown not only by the Mannesmann process of tube-rolling, but by common experience in hollow forging. I unintentionally developed a large central pipe in hammering, in the cold, square bars of even so ductile a substance as copper.

(M) In *Sauveur's* experiment the combined carbon of cast-iron fell from 1.288 to 0.942, or by 0.346 per cent., and the graphite from 3.066 to 1.928, or by 1.138 per cent., on holding at 1,000° for 5 hr. and cooling slowly.[31] It is on the whole probable that most of the carbon which became oxidized in this experiment existed at the time of its oxidation as graphite, and hence that some at least of the 0.346 per cent. of decrease

[28] *Metallographist*, vol. iv., No. 4, p. 319 (Oct., 1901).

[29] *Proceedings of the American Society for Testing Materials*, vol. vi., pp. 230 to 234 (1906).

[30] *Journal of the Iron and Steel Institute*, vol. lxviii. (1905, No. II.), p. 46.

[31] *Journal of the Iron and Steel Institute*, vol. lxxii. (1906, No. IV.), p. 506.

of combined carbon represents a transfer to the graphitic state. The case offers no strong evidence in either direction. I give it only to show that it does not conflict with the present theory.

§ 17. *Phenomena which at first seem to oppose the present or double diagram theory.* Cases in which graphite is changed into cementite by heating at first seem to oppose this theory strongly, because whatever change occurs should be towards greater stability. Cases in which carbon on first entering the solid iron, either from without, as in the cementation-process, or from within, as in the solidification of molten cast-iron, forms cementite rather than graphite do not oppose this theory, because, as already pointed out, the metastable form is usually the one most readily entered. Cases of both these kinds will now be given.

(N) *Cases in which cementation generates cementite instead of graphite.* The familiar fact that the carbon of blister-steel made by the cementation-process exists chiefly in the state of cementite is confirmed by the direct experiments of *Arnold*,[32] whose micrographs of blister-steel show no suggestion of graphite, even when the carbon-content is as high as 1.90 per cent., and of *Brustlein*, who reports that all of the 1.70 per cent. of carbon in a specimen of blister-steel was combined, and therefore must have existed as cementite.[33]

(O) *Margueritte*[34] introduced 6.60 per cent. of carbon in one case and 6.55 per cent. in another into finely divided metallic iron made from the oxalate, by cementation for 3 hr. in carbon monoxide at a low cherry and a bright cherry heat respectively. These color-names correspond to temperatures of 635° and 843° respectively on the scale of White and Taylor; but it is far from certain that these are the temperatures to which Margueritte refers. The fact that the carbon-content of Margueritte's cemented iron was almost exactly that of cementite (theoretically 6.67 per cent.), certainly suggests very strongly that his cementation formed cementite. The case is a very striking one in view of the fact that Margueritte could hardly have

[32] *Journal of the Iron and Steel Institute*, vol. liv. (1898, No. II.), p. 185; Micrograph No. 10 (Plate XX).

[33] Osmond, *Contribution a l'Étude des Alliages*, p. 374 (1901).

[34] *Comptes rendus*, vol. lix., No. 18, p. 726 (Oct. 31, 1864).

suspected the existence of a compound with the formula of cementite.

(P) *Charpy*,[35] confirming Margueritte's results, actually converted two lots of steel filings initially containing 0.09 per cent. of carbon into cementite by heating them at about 650° in potassium cyanide, in one case for 85 hr., in the other for 110 hr. In both cases the steel thus cemented contained 6.72 per cent. of carbon, and therefore must have been converted into cementite, because this is practically the carbon-content of cementite, and because the mass dissolved completely in acid, leaving no trace of graphite.

(Q) *Dr. C. Offerhaus* and I heated powder of pure electrolytic iron, made by Professor Burgess, for 50 hr. in potassium cyanide at about 650°.

Samples taken out at different periods had the following composition:

Time from the beginning, . .	14 hours	31¾ hours	50 hours
	Per Cent.	Per Cent.	Per Cent.
Combined carbon, . . .	5.19	5.74	6.75
Graphite, ...	0.11	2.38	2.88
Total carbon, . .	5.30	8.12	9.63

(R) *Saniter*,[36] by cementing pure iron wire in charcoal at about 900°, gave it a carbon-content of 2.95 per cent., of which 2.42 per cent. was combined.

§ 18. (S) *Cases in which graphite changes into cementite.* A steel with which *Osmond* experimented contained initially 1.38 per cent. of graphite and only 0.24 per cent. of combined carbon. It was therefore equivalent to low-carbon steel contaminated with much graphite, and on heating and cooling underwent only the slight retardations at Ac_1 and Ar_1 characteristic of such steel. On repeating his heatings and coolings the retardations progressively lengthened, and finally became those of a normal high-carbon steel.[37]

Now as the retardation at A_1 is due to the change of carbon back and forth between the states of austenite and pearlite and cannot be caused by graphite, we have here clear evidence that

[35] *The Iron and Steel Magazine*, vol. viii., No. 4, p. 305 (Oct., 1904.)

[36] *Journal of the Iron and Steel Institute*, vol. lii. (1897, No. II.), p. 122.

[37] *Contribution a l'Étude des Alliages*, p. 375–6 (1901).

the graphite was gradually changed into combined carbon, which in the steel when cold would of course exist as cementite.

(T) *Heyn and Bauer*,[38] on reheating slowly-cooled cast-iron of 1.46 per cent. of silicon, 1.32 per cent. of combined carbon, and 2.01 per cent. of graphite, to different high temperatures, holding it there for 30 min., and then quenching it in cold water, found that the combined carbon-content had increased materially. Their results are given in line 17A of Table I., and are condensed in Table V.

TABLE V.—*Change from Graphite into Combined Carbon. From the Data of Heyn and Bauer.*

Temperature to which the cast-iron was reheated before quenching	800°.	900°.	1,180° to 1,200°.
Combined-carbon content after this treatment	Per Cent. 1.52	Per Cent. 1.47	Per Cent. 1.70
Increase of combined-carbon content due to this treatment	0.20	0.15	0.38

(U) *Royston*,[39] starting with malleable cast-iron, all of the carbon of which was graphitic, made 0.85 per cent. of this carbon re-combine at 720°, and 1.50 per cent. of it re-combine at 1,030°, in each case in three hours. See § 53, p. 62.

In § 45, p. 56, another case is given in which graphite seems to have changed into cementite.

§ 19. *The change from graphite into cementite reconciled with the greater stability of graphite than of cementite.* But this change from graphite into cementite is readily reconciled with the theory that graphite is the more stable of the two. In the austenite which exists above A_1, *i.e.*, in regions 4, 5, and 7 of Fig. 1, about 0.90 per cent. of carbon is soluble at A_1, and 2.20 per cent. at 1,130°. When graphitiferous iron is heated above A_1, then, part of this graphite may dissolve in the austenite, until this is nearly or quite saturated with carbon for the existing temperature. When the temperature again falls, and with the fall of temperature the solubility of carbon in austenite decreases, this dissolved carbon is precipitated out of the austenite gradually as the temperature sinks towards Ar_1, at which point the carbon-content should have sunk to 0.90 per cent.,

[38] *Stahl und Eisen*, vol. xxvii., No. 44, p. 1568 (Oct. 30, 1907.)

[39] *Journal of the Iron and Steel Institute*, vol. li. (1897, No. I.), p. 171.

and abruptly on cooling past Ar_1, when this 0.90 of carbon should fall out of solution as the cementite contained in pearlite. But when the carbon thus falls out of solution, it is perfectly natural that it should re-precipitate in the metastable form as cementite, rather than as graphite, especially in view of the pressure which the formation of the bulky graphite would set up. The very first particles of graphite which formed in this re-precipitation would, by their bulk, set up a pressure which would determine that the next particles which precipitated should form cementite. Thus, the transfer of carbon from the graphitic to the cementite state simply implies that part of the carbon has, in heating, re-dissolved in the austenite, and on its re-precipitation in cooling has separated as cementite.

§ 20. *May solution and re-precipitation explain the change from cementite into graphite?* Here some may say that what is sauce for the goose is sauce for the gander. If we complacently explain the change of carbon from graphite into cementite by saying that the graphite has dissolved in the austenite and then re-precipitated in the form of cementite, why may we not answer that the change of carbon from the state of cementite into that of graphite may be explained by assuming that the carbon of the cementite in like manner dissolved in the austenite, and later re-precipitated as graphite? If dissolving in the austenite and then re-precipitating in a new condition explains the change of carbon from graphite into cementite, why may it not explain the change in the opposite direction from cementite into graphite? And if this change may be explained in this way, then it has no force as evidence that graphite is the more stable form.

To this it may be answered that, whereas solution and re-precipitation do explain readily the change from graphite into cementite, they do not explain quantitatively the change from cementite into graphite, which, therefore, is to be referred to the greater stability of graphite. They explain the change from graphite into cementite, because in all of the cases here recorded the quantity of carbon thus transferred is less than that which the austenite is capable of dissolving on heating and giving up again on cooling—viz., 2.2 per cent. The greatest quantity thus changed is 1.50 per cent., reported by Royston, case (U), No. 27 of Table I. (p. 68). But solution and

re-precipitation do not so readily explain the change from cementite into graphite, because the quantity of carbon which thus changes, *e.g.*, 2.35 per cent. in case (B), is greater than the 2.2 per cent. which austenite is capable of thus dissolving. This answer is not in and by itself a very convincing one, and the phenomena of the cementation-process which we will now consider make it even less convincing.

§ 21. *Phenomena of cementation.* The principle just discussed, that the carbon which readily dissolves in austenite up to about 2.2 per cent. naturally re-precipitates as cementite rather than as graphite on cooling, explains easily why the carbon which enters the steel in the cementation-process habitually exists in the cold steel as cementite rather than as graphite, provided that the quantity of carbon in that cementite does not exceed that which the metal can dissolve at the temperature of cementation, as indeed is usually the case. But what is the mechanism by which the cement-carbon content may be carried far above that which saturates the austenite with carbon, as in the cases cited above, in which the steel is converted by potassium cyanide into pure cementite to its very core? And when the cement-carbon content is raised far above 2.2 per cent., as seems to have occurred in Mannesmann's experiments,[40] by heating in charcoal and even in graphite itself, how is the fact that graphite is turned into cementite to a degree far too great to be explained by simple solution and re-precipitation to be reconciled with the theory that graphite is the more stable of the two? M. Osmond gives us a hint from which Dr. Benedicks[41] works out an explanation, which I will now further elaborate.

Once the austenite has become saturated with carbon, at the least cooling and consequent decrease of the solubility of the carbon, part of the dissolved carbon precipitates out as cementite. If the temperature rises ever so slightly, the austenite now exposed to the external cementing graphite is no longer saturated with carbon, and now absorbs additional carbon, which diffuses inwards.

When we first examine this explanation it may seem strained. Picturing to ourselves a piece of steel which has become so far

[40] *Verhandlungen des Vereins zur Beförderung des Gewerbfleisses*, lviii., pp. 38, 40 (1879).

[41] *Metallurgie*, vol. iii., No. 13, p. 439 (July 8, 1906).

carburized that even the central parts contain more carbon than suffices to saturate the austenite, so that even here the austenite contains islands of precipitated cementite, we recognize that, at each rise of temperature, this internal precipitated cementite is sure to compete with the external graphite in the work of supplying to the adjacent austenite the carbon which it is capable of taking because of its increased solvent power for carbon due to that rise of temperature. We ask how, under these conditions, carbon which enters at the outside can by any possibility reach the center of the piece before the demand of that center for carbon shall already have been fully supplied by the local precipitated cementite? By what mechanism, then, can this external carbon raise the carbon-content

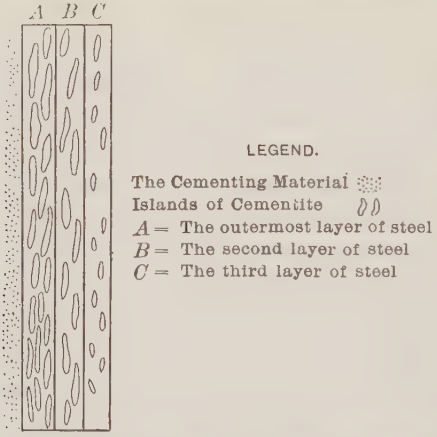
A B C
LEGEND.
The Cementing Material
Islands of Cementite
A = The outermost layer of steel
B = The second layer of steel
C = The third layer of steel

FIG. 7.—THE MECHANISM OF HIGH CEMENTATION.

of the center materially above that which suffices to saturate the austenite?

The answer is a simple one. It is not that at any given rise of temperature any one molecule of carbon rushes from the outside away in to the center, past all the intermediate islands of cementite which are competing with it in the act of carburizing the austenite; but that at each rise there is a wave of carburization surging from the outside to the inside, with but slight centerward translation of the individual molecules of carbon. To understand this easily, consider the three adjoining layers, *A*, *B*, and *C*, Fig. 7, each 0.0001 in. thick, of which *A* is the external layer in contact with the foreign carburizing graphite, and with rather more cementite than *B*,

which in turn has rather more than *C*. At each fall of temperature a little additional cementite is precipitated out of the austenite, to join the islands of cementite already present. At each rise of temperature these islands start to re-dissolve in the austenite. But, because there are more islands in *A* than in *B*, the re-carburizing of the austenite in *A* will be more rapid than in *B*, with the consequence that carbon diffuses from the temporarily more-carburized austenite of *A* into the temporarily less-carburized austenite of *B;* and in like manner carbon diffuses from the austenite of *B* into that of *C*, and so on from the very outside to the very core. Thus there is a wave of carburization which may travel some inches, though the individual molecules of carbon may travel only a fraction of a 10,000th of an inch. At the same time the external graphite competes with the cementite islands of *A* in the work of re-carburizing those microscopic particles of austenite which lie at the outside of *A* in contact with that graphite; and thus it comes about that some carbon enters the piece from the external graphite. By repetition of these waves of carburization and this slight uplicking of carbon from the external graphite it comes about that, after a sufficient number of even extremely slight oscillations of temperature, the carbon-content of the very center may be brought far above that needed to saturate the austenite.

§ 22. *Summary.* However satisfying this explanation of the genesis of cementite in the cementation-process may be, the very facts (1) that extraneous graphite thus changes so abundantly into cementite which persists so well, and (2) that even indigenous graphite may thus change into cementite (cases S, T, and U of § 18), show that the mere fact of the change from cementite into graphite in the many cases given (cases A to M, of § 16) is not in itself strong evidence that graphite is the more stable of the two. Nor is there strong evidence to this effect in the fact pointed out in § 20, that the formation of cementite from graphite is explained quantitatively by solution and reprecipitation, whereas the reverse formation of graphite from cementite is not so explained, because the quantity of graphite which thus forms exceeds the quantity of carbon which austenite can dissolve. The force of this fact is lessened by the explanation just given of the change of a still

greater quantity of extraneous graphite into cementite in the cementation-process; because this leads us to suspect that, under favoring conditions, the quantity of indigenous graphite which can be changed thus into cementite may exceed the solubility of carbon in austenite. If the change of extraneous graphite into cementite may thus exceed that solubility, why may not that of indigenous graphite? Indeed, the suspicion just referred to is greatly strengthened by the cases examined in §§ 45 and 53, pp. 56 and 62, because in these cases the quantity of carbon which has changed from the graphitic to the cementite state seems to be greater than that which the austenite can dissolve at the temperature actually reached.

Thus the evidence of the greater stability of graphite than of cementite is to be found, not in such facts taken in their nakedness, but rather in the attendant conditions. These are:

1. That the change from cementite into graphite is exothermic, and hence in the direction of greater stability.

2. That this change does take place in spite of the evident opposition to it which the bulkiness of the resultant graphite sets up, through the pressure which its formation must cause.

3. That cementite is habitually the form first assumed by the carbon of solid iron, whether it comes from within, as in the solidification of molten iron, or from without, as in the cementation-process; and from the decomposition of this cementite comes part and often the whole of the graphite which eventually forms. This is the habitual order of things, the formation of the metastable first, and of the stable later.

4. That, according to Benedicks' keen observation,[42] whereas there is much microscopic evidence tending to show that cementite often changes into graphite directly without passing through solution in austenite, yet there is no such evidence of a direct change from graphite into cementite. Prof. William Campbell's observations and my own tend to confirm this.

5. That in the regular manufacture of malleable castings, the shifting of carbon from the initial state of cementite into that of graphite is not only certain, every day, in every batch, and one might almost say in every part of every piece treated, but further apparently often exceeds in quantity the percentage (2.2 per cent.) which suffices to saturate the austenite.

[42] *Metallurgie*, vol. iii., No. 13, p. 433 (July 8, 1906.)

Of these five, the first three seem to me very cogent, while the contributory weight of the last two is considerable.

Part II.

§ 23. *Topography of the graphite-austenite diagram.* (A) *The freezing-point curves.* Charpy[43] found that the cementite-austenite eutectic freezing-point was very slightly lower than that of the graphite-austenite eutectic, but that the difference was only about 10° or 15°. This agrees with Le Chatelier's prophecy on this subject, as shown by the relative position of these two eutectic freezing-points in the Roberts-Austen[44] diagram of 1899. But Charpy's determination does not seem to have been very positive.

§ 24. (B) *The transformation-curves.* Just as, when the red- or white-hot iron consists of a mixture of austenite and cementite, the solubility of carbon in the austenite diminishes as the temperature falls, as represented by the curve Ar_3, *Sa*, of Fig. 1; so it is but natural to assume that, when the metal consists of a mixture of graphite and austenite, there shall be a like decrease in the solubility of carbon with falling temperature, which can be represented in the graphite-austenite diagram by a line analogous to Ar_3. Further, it is natural to assume that the solubility of carbon in austenite when the phase presented to the austenite is graphite differs from the solubility when that phase presented is cementite.

There has been an important attempt to determine the curve of solubility of graphite in austenite by means of the data of Charpy and Grenet, and the results of this attempt are said to be confirmed by the earlier data of Mannesmann. After explaining the Charpy-Grenet procedure, I will discuss the solubility-curve which has been based on their results, then offer a curve which agrees more closely with them, then discuss the competence of these results as evidence of the position of this solubility-curve, and then consider how far Mannesmann's data actually confirm the curve deduced from the Charpy-Grenet results. After summarizing this discussion, I will next

[43] *Comptes rendus*, vol. cxli., No. 23, p. 950 (2e semestre, 1905).

[44] *Fifth Report to Alloys Research Committee*, read before Institution of Mechanical Engineers, Feb. 9, 1899, Fig. 10, Plate 4, *Proceedings of the Institution of Mechanical Engineers* (1899).

consider what inferences can be drawn from the other data at hand.

§ 25. *The Charpy and Grenet line* is shown in Fig. 8 by the points r, r', and r'', which correspond to the following numbers:

	r	r'	r''
Temperature, Centigrade, ...	800°	900°	1,000°
Solubility of carbon, per cent., . . .	0.70	0.90	1.30

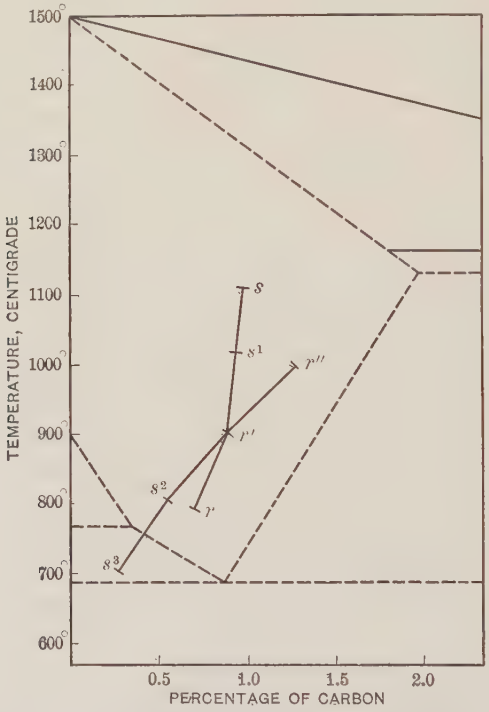

FIG. 8.—SOLUBILITY OF CARBON IN AUSTENITE IN PRESENCE OF GRAPHITE, TOGETHER WITH PARTS OF THE STABLE AND METASTABLE DIAGRAMS FROM FIG. 4.

Their general procedure was, first, to melt cast-iron and granulate it in water, so that all or nearly all of its carbon was initially either combined as cementite or dissolved in the austenite; then to reheat it to a given high temperature, Tn, and hold it there for a long time with the intention, first, of changing the carbon of the cementite into graphite, and, second, of enabling the carbon initially dissolved in the austenite to fall out of solution as graphite until the remaining austenite was just saturated with carbon; then to quench the iron in cold water

in order to prevent any appreciable further separation of graphite during cooling, and thus to preserve in solution the quantity which had actually remained in solution at Tn; then to determine the total carbon and also the graphite; and, finally, to find by difference the carbon which had remained in solution in the austenite. In certain cases the formation of graphite was started by heating the metal close to the melting-point; the metal was then cooled slowly to Tn, and then quenched, either immediately after reaching Tn or after staying there for a long time.

Without here passing upon the competency of the data to prove the solubility of graphite in austenite, we may here note that the numbers which Charpy and Grenet get by difference are simply the non-graphitic or agraphitic carbon, which includes both that dissolved in the austenite and any present as cementite; and, further, that even the dissolved carbon, though in spots it is that dissolved in presence of graphite, in other spots would be that dissolved in presence of cementite, if any cementite were present. But let us consider this question later, and here ask only what the results of Charpy and Grenet show on their face.

§ 26. *Discussion of the points r, r′, and r″ of the Charpy and Grenet line.* All the results which Charpy and Grenet reached in this investigation are given in lines 1 to 5 of Table I., p. 68.

§ 27. *Point r′, solubility at* 900°. All seven determinations of this point made with cast-iron containing either 1.25 or 2.10 per cent. of silicon agree surprisingly. The higher numbers reached with the low silicon-content of 0.80 per cent. are satisfactorily explained away by the great lag due to the smallness of this silicon-content, so that there is no reason here for doubting the position assigned to this point, 0.90 per cent.

§ 28. *Point r″, solubility at* 1,000°. I do not understand why the position 1.30 per cent. has been assigned to this point. It is clear that the cast-irons initially contained much more combined carbon than is soluble in the austenite, and that, during the exposure to 1,000°, the sum of the cementite-carbon and of the dissolved carbon was decreasing towards the solubility or saturation-point of carbon in austenite. This decrease might or might not bring the total agraphitic-carbon content fully down to the saturation-point; but it certainly could not bring

it below that point. Hence, the lowest result must be taken simply as that which is least in excess of the saturation-point, and hence is the closest approximation to the truth. I cannot see why the number 0.93 per cent. found with iron No. 5 at 1,000° should not be accepted, especially because it is supported in a measure by the number 0.98 per cent. found for iron No. 4 at 1,000°.

§ 29. *Point r, solubility at* 800°. The position assigned, 0.70 per cent., I cannot accept. First, I find no such result recorded. Second, I see no reason for rejecting the number 0.54 found for iron No. 5 at 800°. Indeed, the number 0.43 per cent. found for No. 3 seems to me practically as trustworthy as any of these. This particular specimen, after very slow cooling to 900°, was then cooled completely, and reheated the next day to 900°, and then cooled in 2 hr. to 800°, then held for 2 hr. at 800° and quenched. To this it has been objected that, in cooling below 800°, the carbon-content of the austenite may have fallen below the 800° saturation-point, and this carbon thus rejected may not have been re-absorbed the next day. It seems a fair answer that the austenite had 2 hr. above 800°, and 2 more at 800°, to re-absorb this carbon, and that I know of no evidence that the re-absorption of carbon is less rapid than its expulsion. In this particular case any carbon expelled below 800° would remain very close to the austenite whence it was expelled, because of the slowness of diffusion at these low temperatures, and because it should lie so close it should be re-absorbed quickly. Personally, then, I should adopt 0.43 per cent. as the most probable number; but to avoid all appearance of unfairness to the numbers which I am criticising I adopt 0.54 per cent.

§ 30. *A more reasonable Charpy and Grenet line.* If we plot the points found in this discussion to be the nearest approximations to the truth, 0.90 per cent. for 900°, 0.93 per cent. for 1,000°, and 0.54 per cent. for 800°, and proceed in the same way to select the nearest Charpy and Grenet approximations for 700° and 1,100°, we get the line s, s^1, r', s^2, s^3 in Fig. 8. Personally, I attach little importance to this line, for reasons which I will next explain. But at least it has the merit of being in harmony with the results on which it is based. These numbers are given in Table VI.

TABLE VI.—*A More Reasonable Charpy and Grenet Line.*

Temperature	1,100°	1,000°	900°	800°	700°
Solubility	1 per ct.	0.93	0.90	0.54	0.28
Designation letter	s	s^1	r'	s^2	s^3

§ 31. *How valid evidence do the Charpy and Grenet results give as to the solubility of graphite in iron?* We have now seen that not the line r–r'' but the line s–s^3 is, in fact, indicated by these results. Let us next ask as to the competency of the evidence which these results give, so that we may know what weight to attach to this line s–s^3.

§ 32. *Definitions.* To facilitate discussion let us, for the purposes of the present paper only, adopt these terms.

Tn = any given temperature for which we are seeking the solubility of carbon in austenite in the presence of graphite, *i.e.*, the saturation-point.

STn = the percentage of dissolved carbon which thus just saturates the austenite at Tn, *i.e.*, the solubility of carbon in austenite at Tn. In the same way let us call ST 1,000° the solubility at 1,000°, etc.

AGr = the total agraphitic or non-graphitic carbon-content, including the dissolved and the cementite-carbon.

§ 33. *Discussion.* What we are seeking is the solubility of graphite in pure, *i.e.*, silicon-less iron; what the Charpy-Grenet numbers give is the total agraphitic carbon actually present in very siliciferous iron. Those who have accepted these numbers as equal to that solubility have tacitly assumed that AGr = STn, and in doing this have made three separate assumptions:

(A) That the silicon-content does not affect the solubility;

(B) That the agraphitic carbon is all dissolved carbon; and

(C) That this dissolved carbon actually saturates the austenite at Tn.

With regard to assumption (A), it is of course admitted that, within limits, the more silicon is present the more readily does carbon pass out of combination as cementite or out of solution in austenite into the state of graphite; but it is held by some that the action of silicon is purely catalytic, not changing the true solubility of carbon in austenite, but only lessening the lag, and thus hastening compliance with that solubility, or, in other words, hastening the separation as graphite of any carbon

actually retained in solution by lag, in excess of the true solubility or saturation-point,[45] and also hastening the change from the metastable cementite to the stable graphite. Neither line, r, r', r'', nor s, s^3, has the least claim to accuracy unless this assumption is made, because every point in each line rests upon one or more determinations made with iron containing an important quantity of silicon. In § 43 the influence of silicon is discussed.

Assumption (B) is that of the three forms of carbon: (1) carbon combined as cementite, Fe_3C; (2) carbon dissolved in the austenite; and (3) free graphite, only the second and third are present, so that the whole of the great quantity of cementite which, because of the initial absence of graphite, we infer must have been present initially, has disappeared, either changing into graphite or dissolving in the austenite. It is only on this assumption that either r, r'' or s, s^3, or any other curve of the solubility of graphite in austenite, can be based on the Charpy-Grenet results. If any cementite is present, then the agraphitic carbon, instead of equaling the carbon dissolved in the austenite in the presence of graphite, as has to be assumed in order to infer the solubility of carbon in austenite in presence of graphite from the Charpy-Grenet data, really represents three distinct things: (1) carbon present as cementite; (2) in certain microscopic spots carbon dissolved in austenite in presence of that cementite; and (3) only in certain other microscopic spots, that which we are seeking, the carbon dissolved in austenite in presence of graphite.

Assumption (C) is that the austenite, which is justly assumed to have been supersaturated with carbon when first cooled to Tn, has, in the stay at Tn, actually expelled all carbon in excess of the true saturation-point, so that the combined carbon found actually represents the saturation-point, or the true solubility of carbon in austenite in the presence of graphite at Tn.

To assumptions (B and C), taken jointly, it seems a reasonable answer that, if the experience of centuries with the process of annealing malleable castings has shown that two days at full

[45] . . . "die Graphitbildung wird durch steigenden Si-Gehalt stark beschleunigt, aber der Endzustand wird durch Si nicht beeinflusst, wie besonders die Kurven für 1.20 % Si und 2 % Si bei 900° lehren." Benedicks, *Metallurgie*, vol. iii., No. 13, p. 432 (July 8, 1906).

heat are needed to bring about even the incomplete change from cementite into graphite, which suffices for the commercial needs of the process, and if even then an extremely slow cooling is needed to make the content of carbon dissolved in the austenite fall near enough to the saturation-point for commercial purposes, it is hardly safe to assume that the 4-hr. periods of Charpy and Grenet have sufficed to do these same things with the extreme thoroughness needed for scientific purposes, in default of direct evidence that they have sufficed. Further, we have seen in § 18 how readily the carbon, even when it has once become graphitic, shifts back into the cementite state. Hence I infer that assumption (B) is very rash.

Thus all three of these necessary assumptions are at least questionable.

Now any error in assumption (A) tends to bring the Charpy-Grenet results below the true solubility; and any error in either assumption (B) or (C) tends to bring those results above the true solubility. Hence the solubility, instead of being equal to those results as has been assumed, may, because of the errors in these assumptions, be either greater or smaller than those results, and by an unknown and indeterminate amount.

§ 34. *Mannesmann's results.* In confirmation of the trustworthiness and accuracy of the position of the line r, r', r'', it has been asserted that, representing as it does the results of Charpy and Grenet, it is confirmed by those of Mannesmann reached in a wholly different manner.[46] That Mannesmann's manner was wholly different is true, but so far as I can see his results not only do not confirm those of Charpy and Grenet, but do not bear on them in the remotest possible way.

In trying to test the truth of the then general opinion that the carbon-content could not be raised above 2 per cent. by cementation without melting the resultant cast-iron, Mannesmann found that, by cementing wrought-iron in charcoal, he could make not only steel and white cast-iron of any and every carbon-content up to 4.76 per cent., but also both light-gray and dark-gray cast-iron, and in every case wholly without fusion,

[46] "Von grossem Interesse ist, dass die drei Löslichkeitswerte des Graphits, die man aus den schon erörterten Bestimmungen Charpy und Grenets gewinnt, mit denjenigen auf ganz anderem Wege von Mannesmann erhaltenen vorzüglich übereinstimmen." Benedicks, *Metallurgie*, vol. iii., No. 13, p. 435 (July 8, 1906).

whence he inferred that as much carbon can be introduced by cementation as by fusion.[47] He then sketched a curve, which I here reproduce as AB in Fig. 9, showing, first, how the percentage of carbon which can thus be introduced increases with the temperature, and, second, that the temperature needed to raise the carbon-content above, say, 4.75 per cent., is so near the melting-point of the resultant cast-iron (as shown by the melting-point curve CD) that it is extremely difficult to carry the carburization farther without melting the iron.

§ 35. *Mannesmann's curve a freehand sketch.* All the internal evidence tends to show that this, his curve AB of the attain-

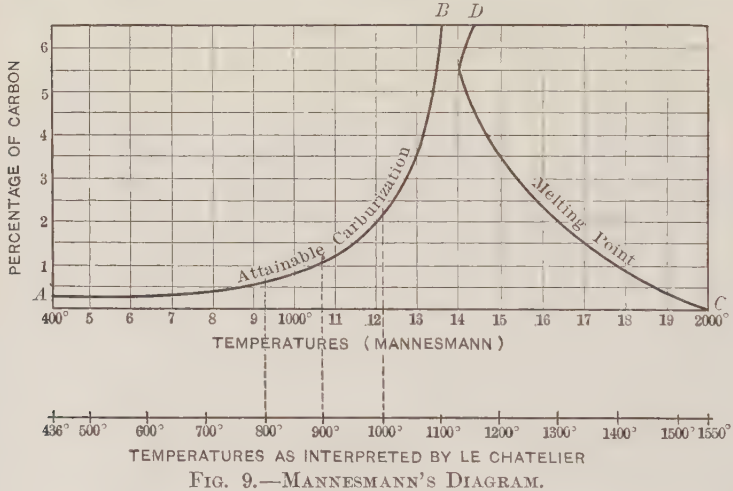

FIG. 9.—MANNESMANN'S DIAGRAM.

able degree of carburization, does not pretend to be anything more than a sketch. In the first place, there are no numerical measurements of temperature, nor, indeed, estimates of temperature in the text. He refers only to "yellow heat," "light-yellow heat," and other like color-names; he never in the whole paper gives a single temperature in degrees, and he never attempts to identify the temperature of any of his experiments by saying that it was the melting-point of some known substance, or in any like way.

In the second place, the perfect smoothness of his curve, without a single dot, spot, or cross to indicate the result of any

[47] *Verhandlungen des Vereins zur Beförderung des Gewerbfleisses*, lviii., p. 31 (1879). This very valuable paper was written by Mr. Mannesmann at a very early age.

one determination, in itself goes to show that this curve was only a sketch, and made no pretense to accuracy.

In the third place, though his actual results seem to have reached only to 4.76 per cent. of carbon, his curve itself runs up to 6.5 per cent. (misnumbered in the original), so that the part between 4.76 and 6.50 per cent. seems to be extrapolated, and therefore, of course, nothing but a fancy sketch. But this extrapolated part is of exactly the same smoothness and general appearance as that below 4.76 per cent., so that it is not safe to assume that this latter part is anything more than a fancy sketch.

In weighing evidence it is a fundamental rule that the burden of proof of validity is on him who presents the evidence. No such proof is here offered, and in view of the *prima facie* indications that the curve is nothing but a sketch, it should, in my opinion, be accepted only as a sketch.

§ 36. *Mannesmann's curve does not agree with that of Charpy and Grenet.* If, for argument's sake, the curve is accepted as accurate and trustworthy, it still fails to confirm the Charpy and Grenet curve, for the excellent reason that it differs radically from it, as is seen on comparing lines 1 and 3 in Table VII.

TABLE VII.—*Comparison of the Charpy and Grenet Data with Those Attributed to Mannesmann.*

Temperature	800° Per Ct.	900° Per Ct.	1,000° Per Ct.
(1) Results claimed from Charpy and Grenet	0.70	0.90	1.30
(2) Results from Charpy and Grenet as modified by the present writer	0.54	0.90	0.93
(3) Results from Mannesmann (A) using his temperature-scale	0.45	0.55	0.75
(4) (B) Applying Le Chatelier's correction[a] to Mannesmann's temperature-scale	0.60	1.05	2.15

[a] *Le Chatelier's Correction.*—In order to make allowance for Mannesmann's evident errors in temperature, Le Chatelier (*Contribution a L'Etude des Alliages*, p. 382), changes Mannesmann's melting-point of pure iron to 1,550°, assumes that his 500° is accurate, and divides the intermediate temperatures up proportionally.

If it is pleaded that some correction should be made in the temperature to allow for Mannesmann's evidently wrong temperature-scale, the question immediately arises, what correction? If we adopt that proposed by Le Chatelier, we still find that Mannesmann's results do not agree at all with those of

Charpy and Grenet, as is shown by comparing lines 1 and 4 of Table VII.

§ 37. *Mannesmann's curve does not represent solubility.* Finally, the objection which lies against Charpy and Grenet's results—viz., that they do not at all represent the carbon dissolved in the austenite, because their method of chemical analysis was such that any carbon present as cementite would be included, lies far more damagingly against Mannesmann's, for a simple reason. Any arguments brought up to persuade us that in Charpy and Grenet's experiments the austenite was really saturated with carbon and that no cementite was present, and hence that their results really represent the solubility of carbon in the presence of graphite, would be without force if applied to Mannesmann's results, because his conditions were such that he either might not saturate his austenite with carbon, or, if he did saturate it, then that he would go right on adding carbon in the form of cementite, and not in that of graphite. These conditions were that he cemented wrought-iron rapidly at various unstated temperatures, assumed to include 800°, 900°, and 1,000°, without attempting to start the formation of graphite by raising the temperature intentionally above that at which the cementation took place. Such rapid cementations, while on one hand they may not saturate the austenite, yet on the other hand habitually lead to an abundant formation not of graphite but of cementite, as in the common course of the cementation-process exemplified in cases O, P, Q, and R of § 17, p. 29. If any fact in this whole field is abundantly and clearly proved it is this. Indeed, my original reason for naming this substance "cementite" was that it contains Rinman's "cement carbon," so-called because it is characteristic of cemented steel. M. Charpy himself has shown that iron may be converted into a mass of pure cementite by thus cementing long enough with potassium cyanide. The very fact that among the products of Mannesmann's experiments were "the most varying grades of white cast-iron with from 2 to 4.76 per cent. of combined carbon" (*amorphem Kohlenstoff*, p. 39, l. 4) shows that his conditions led to the formation of much cementite, for a white cast-iron rich in carbon is necessarily rich in cementite. He made no attempt to distinguish between dissolved and cementite carbon, if indeed he knew that they were distinct things.

§ 38. *Summary tof he discussion of the accuracy of the Charpy and Grenet curve* r, r', r''. There are the following reasons, each sufficient in itself, for rejecting this curve as representing the solubility of carbon in pure austenite in the presence of graphite.

Not r, r', r'', but s, s^3, is the curve which the data of Charpy and Grenet indicate.

Neither curve deserves confidence, because of the probably large errors introduced by two and perhaps three unjustified assumptions, two of which tend to make the apparent solubilities greater than the true, while the third tends in the opposite direction, so that the true curve may lie an indeterminate distance on either side of the apparent curves thus formed.

Three reasons, each sufficient in itself, show that Mannesmann's curve does not confirm the curve r, r', r''.

(1) It is very far from r, $r,'$ r'', even if corrected for his wrong temperature-scale.

(2) It cannot be taken as anything more than a freehand sketch, without pretense to quantitative accuracy.

(3) It includes, along with the dissolved carbon which r, $r,'$ r'' is supposed to represent, a quantity of cementite-carbon which almost certainly must have been large.

In my opinion, this Charpy and Grenet curve should be rejected outright as absolutely without any basis whatsoever.

§ 39. *What inferences can be drawn from our present data as to the solubility of graphite in austenite?* In the attempt to answer this question I have condensed in Table I. all the data which seem to bear on it. Before going farther let us recognize that there are four distinct sets of numbers with which we have to do. See § 32, p. 41.

AGr, the percentage of agraphitic or non-graphitic carbon, usually but inaccurately called the total "combined" carbon. This is inaccurate, because in suddenly-cooled metal part of this carbon is not combined, strictly speaking, but dissolved.

DC, the percentage of dissolved carbon.

Cem. C, the percentage of cementite-carbon; and

STn, the solubility at any given temperature.

Now it is STn that we are seeking, but it is only AGr that is ever reported. As to the quantity of Cem. C we are much in the dark. As to the relations between these different num-

bers, so far as I now see all that can be asserted positively is given in the following expressions:

(1) AGr = DC + Cem. C, hence

(2) DC = or < AGr.

If, in any given experiment, DC has increased at a given temperature, Tn, then the final DC cannot be greater than STn, or

(3) DC = or < STn.

But if, in such experiment, DC has decreased, then the final DC cannot be less than STn, or

(4) DC = or > STn.

(3) and (4) simply express the law that any change in the dissolved-carbon content, DC, must carry it towards equilibrium, *i.e.*, towards STn, but not beyond STn.

We might be tempted to go beyond this and assume that whatever change has taken place in the quantity of cementite in any experiment must be in the direction of greater stability, and therefore must be a decrease, representing a change from cementite into graphite, to be represented thus:

(5) Cem. C initial = or > Cem. C final.

But the instances of the rapid change of graphite into cementite in § 18 show that this step would be most unsafe.

If we knew DC our task would be comparatively easy; but the attempt to infer DC from the changes in AGr without any positive knowledge as to Cem. C is very difficult.

The chief difficulties in the way of bringing order out of this chaos are,

(A) our uncertainty as to whether silicon does or does not lessen STn;

(B) The uncertainty as to how much Cem. C is present, and hence as to the amount by which AGr exceeds DC;

(C) The uncertainty as to whether, in any given case, DC equals, or exceeds, or is less than STn.

Those who have published curves of solubility may not have realized the seriousness of these difficulties.

§ 40. *Cases in which AGr, made high by long exposure to a higher temperature, Th, decreases on exposure to a lower temperature, Tl.* Here the probability that at Th DC has risen above STl makes it probable, from (4), that DC after holding at Tl is at least as great if not greater than STl, or DC = or > STl.

But, from (2), AGr = or > DC. Hence in such cases

(6) STl = or < AGr.

In such cases, then, the solubility is at least as small as the least trustworthy value found for AGr, provided, of course, that the AGr reported is that which actually existed at Tl. This may be assumed to be true when the iron is quenched at Tl, but not when, as in the experiments of Wüst and Schlösser, Table I., lines 6 to 14, the iron is cooled slowly from Tl down, because in this slow cooling DC may fall below STl for the low temperature in question, Tl, and hence AGr found after this slow cooling may be less than that at Tl.

§ 41. *Cases in which AGr, made low by long exposure to a lower temperature, Tl, increases on exposure to a much higher temperature, Th.* After long exposure to a temperature like 750°, which, though relatively low, is yet high enough to give a degree of freedom of molecular movement sufficient to permit any excess of DC to begin falling out of solution, so that DC approaches STl, the probability that DC then is less than STh makes it probable, from (3), that after holding at Th DC will be at most as small if not less than STh, or

(7) DC = or < STh.

If in such case Cem. C = 0, then DC = AGr, and hence from (7),

(8) STh > or = AGr.

In such cases the solubility is at least as great as the greatest value found for AGr. But unfortunately we are not sure in any of these cases that Cem. C = 0. Indeed, it may be large.

If Cem. C is large, *e. g.*, if Cem. C > STh — DC, then

(9) STh < AGr.

For instance, if the solubility was 1.00 per cent., the carbon actually dissolved 0.80 per cent., and the cementite-carbon 0.30 per cent., the total agraphitic carbon would be 1.10 per cent., or greater than the solubility.

By comparing (8) and (9) we see that the solubility in this case may be either greater or less than the agraphitic carbon, or

(10) STh >, = or < AGr.

The assumption on which (9) rests, that even after a long exposure to a high temperature, much of the metastable cementite may remain unconverted into the stable graphite, seems to me wholly reasonable. Indeed, carbon which has

once changed into graphite may change back indirectly into cementite, as is shown clearly by Osmond's experiment, case S of § 18, p. 30. The fact that the cooling-curve of this metal was that of a high-carbon steel, with its Ar_1 very strongly marked at 700°, shows not only that the originally very small quantity of AGr (0.24 per cent.) had increased very greatly, but further that, on cooling past Ar_1, much of that carbon actually changed from DC into Cem. C in the form of pearlite. Benedicks [48] accepts this belief that graphite may thus change into cementite. § 18 gives some additional cases.

§ 42. *To sum these two cases up*, if the iron has been cooled from a much higher to a much lower temperature and then quenched, the solubility is at least as low as the lowest trustworthy value then found for the total agraphitic carbon; but if the iron has been heated from a lower to a much higher temperature and then quenched, the solubility at that higher temperature may be either greater or less than the greatest trustworthy value found for the agraphitic carbon, and by an indeterminate amount. In short, if we call the temperature at which the iron is quenched TQ, and STQ the solubility at TQ, then on quenching after falling temperature

(11) STQ = or < AGr;

whereas on quenching after rising temperature

(12) STQ >, =, or < AGr.

Indeed, we may go farther and enunciate the law as follows:

If before quenching DC has been decreasing, then

(11) STQ < or = AGr.

But if DC has been increasing, then

(12) STQ >, =, or < AGr.

In short, for the foregoing reasons, cases like those before us in which the quantity of cementite is unknown, are competent only to give an upper limit for the solubility of carbon; that is to say, only to give a number which the solubility cannot exceed, though it may be less than that number by an indeterminate

[48] "Hingegen lässt sich ja Cementit aus Ferrite oder Mischkristallen + Graphit auf Umwegen sehr einfach darstellen, wenn nur erhitzt wird, so dass der Graphit in Lösung geht. Beim Abkühlen wird sich die Kohle zunächst als Cementit abscheiden. Durch genügend hohes Erhitzen eines jeden stabilen Fe-C-Systems und genügend schnelles Abkühlen lässt es sich in das metastabile System überführen." C. Benedicks, *Metallurgie*, vol. iii., No. 14, p. 473 (July 22, 1906).

amount. They are in their very nature incompetent as evidences of the lower limit for solubility.

But the matter is even worse than this. Because cementite may re-form on heating even graphitiferous iron, such irons may thus be conglomerates in the second degree, made up of independent conglomerates (A) of austenite and graphite, and (B) of austenite and cementite. Hence, if the solubility in such irons were actually determined, it would not be that here sought, viz., the solubility of carbon in the presence of graphite, but a number somewhere between this and the solubility in the presence of cementite, and therefore presumably greater than the solubility in presence of graphite alone.

To sum this up, the present evidence in its nature can give us only an upper limit for solubility; and even the solubility thus vaguely pointed at is not the solubility sought, viz., that in the presence of graphite, but is probably a greater number somewhere between the solubility sought and the solubility in the presence of cementite.

Here, as usual, the work of the fingers is useless until interpreted by the mind. It is only the raw material out of which the mind must construct the truth.

Nevertheless, let us go on to see what upper limit can be set from the data at hand.

§ 43. *Does the presence of silicon lessen the solubility of carbon?* This question ought, if possible, to be answered before we attempt to determine the solubility. If silicon does not lower STn, then the results reached with irons of different silicon-content can be used to check and verify each other. If it does lower the solubility, then the position of the curve of solubility shifts with the silicon-content. If, finally, we are in doubt whether silicon does or does not lower STn, then this same doubt attaches to the inferences which we reach from a comparison of the results given by irons of different silicon-content.

When the numbers of Table I. are first examined they promise to give an answer to this question, but in view of the considerations which have just been set forth in §§ 39 to 42, that answer proves to be equivocal.

Passing over the general decrease in agraphitic-carbon content with increasing silicon in cases 1 to 5 and 6 to 14 of

TABLE VIII.—*Data of Table I. Rearranged.*

(The designation numbers are those of Table I.)

Desig. No.	Si.		AGr (C.C.)	Tn°.	Time.	AGr (C.C.)	Tn°.	Time.	AGr (C.C.)	Tn°.	Time.	AGr (C.C.)	Tn°.	Time.	AGr (C.C.)	Tn°.	Time.
5	2.10	Charpy and Grenet. On reheating a cast-iron initially containing 3.10 per cent. C. C., and then quenching it, the AGr (C. C.) had fallen to.....	0.54	800°	4 hr.	.....	.....	.....	0.88 0.90 0.90	900°	4 hr.	0.93	1,000°	4 hr.	1.00	1,100°	4 hr.
18 A	4.23	Heyn and Bauer. On reheating a graphitic cast-iron of 4.23 per cent. silicon and quenching it, the AGr (C. C.) fell from between 0.59 and 0.84 per cent. to	0.19 to 0.69	800°	½ hr.	.....	.....	.....	0.62 to 0.67	900°	½ hr.	0.53 to 0.82 0.53	1,000°	½ hr. 2 hr.	0.56 to 0 78	1,050°	½ hr.
19	0.10±	Charpy. On reheating a cast-iron of 0.10± per cent. silicon and quenching it, the AGr (C. C.) changed from 0.41 per cent. to.....	0.31	750°	6 to 7 hr.	0.85	850°	6 to 7 hr.	.....	....	.....	1.08	1,000°	3 hr.	1.36	1,050°	3 hr.
20	0.10±	Charpy. Cast-iron containing 2.82 per cent. AGr (C. C.) initially was reheated to 1,150°, cooled slowly to 1,000° and quenched.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.92	1,000°	.....	.....	.....	.....
21	0.10±	Charpy. Cast-iron like No. 20 was cooled from the molten state to 1,000°, held there 3 hours and quenched.	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.95	1,000°	3 hr.	.....	.....	.....

No. 5. Charpy and Grenet heated to the temperatures indicated a cast-iron which contained originally

Carbon.						
Per Cent. AGr.	Per Cent. Gr.	Per Cent. Total.	Silicon. Per Cent.	Manganese. Per Cent.	Phosphorus. Per Cent.	Sulphur. Per Cent.
3.10	0.20	3.30	2.10	0.12	0.01	0.02

and then quenched it in cold water.

No. 18A. Heyn and Bauer, *Stahl und Eisen*, vol. xxvii., No. 44, pp. 1567–9 (Oct. 30, 1907); their iron No. 419. The iron was made by melting together cast-iron and ferrosilicon of the following composition:

	Carbon. Per Cent.	Silicon. Per Cent.	Manganese. Per Cent.	Phosphorus. Per Cent.	Sulphur. Per Cent.
Cast-iron, ...	2.95	0.63	0.22	0.09	0.33
Ferrosilicon, ...		96.	0.27	0.041	0.108

Thus the cast-iron must have contained much sulphur.

No. 19. Charpy, *Revue de Métallurgie*, vol. v., No. 2, Mémoires, p. 77 (Feb., 1908). This case is the same as No. 19 of Table I.

Table I., because it can readily be explained by catalytic action, I have condensed in Table VIII. certain cases which cannot be so lightly disposed of.

Here the evidence tending to show that silicon does not lessen the solubility is certainly striking. It is that the great difference in silicon-content between case 4 and case 5, and that between case 5 on one hand and cases 20 and 21 on the other, leads to no material difference in agraphitic-carbon content, as shown in the former group by the agraphitic carbon found at 900°, and in the latter group by that found at 1,000°. The evidence of the former group is all the more striking because of the large number of results which agree almost exactly, 3 out of 4 in case 4 and all three in case 5. But on closer examination I brush both these agreements aside as simply two of the many cases in which accidental coincidence obstructs the progress of knowledge. My reason for this apparently rash step is that these coincidences really contradict each other. Thus, following the law that the lowest trustworthy result when AGr is decreasing should be taken as that which least exceeds STn, I take STn for case 5 as follows:

TABLE IX.—*Most Probable Apparent Values for the Solubility of Carbon.*

Temperature	700°		800°		900°		1,000°		1,100°
STn not greater than	0.28		0.54		0.88		0.93		1.00
Difference		0.26		0.34		0.05		0.07	

Thus, I assume that STn must be at least as small as 0.54 per cent. for 800°, as 0.93 per cent. for 1,000°, and as 1.00 per cent. for 1,100°. Admitting this, and further admitting that it is extremely probable that STn from 700° to 1,100° is a smooth curve, I hold that ST 900° is extremely probably less than the 0.88 per cent. reported, because this number falls too little short of the number (0.93 per cent.) found for 1,000°. Hence, I hold that the agreement of 6 out of 7 determinations in cases 4 and 5 at 900° is simply an accidental agreement in error. So, too, I hold that the value, 0.93 per cent., found for STn at 1,000° in case 5 is very probably too great, because it falls too little short of the number, 1.00 per cent., found for

1,100°, and because it greatly exceeds the number, 0.53 per cent., found by Heyn and Bauer at 1,000°, in case 18A. In short, these agreements between the agraphitic carbon of cast-irons which differ greatly in silicon cannot be accepted as valid evidence that silicon does not influence the solubility of carbon, because in each of these agreements the reported agraphitic carbon of the high-silicon iron is probably much greater than the dissolved carbon proper.

Further evidence may show that the numbers which I thus reject are right. Nevertheless, my present position seems to me the only logical one, to reject numbers the only value of which lies in their agreement, when they are contradicted by other numbers intrinsically more probable because of the law that, when AGr is decreasing, the STn must be as low as the lowest trustworthy AGr reported.

As between this law on one hand, and an agreement between results which have nothing but that agreement in their favor on the other hand, the former seems to me a firmer basis for probability.

Evidence that silicon does lower the solubility must, unless based on knowledge of the quantity of cementite present, be inconclusive in the very nature of the case. Without that knowledge the evidence would have to be both extensive and harmonious to be convincing. Just as we pass by the evidence of the two important groups, cases 1 to 5 and cases 6 to 14, because in each the decrease of agraphitic carbon-content with increasing silicon can be explained by the assumed catalytic action of that silicon, so may we pass by the very striking results in the high-silicon iron of cases 18 and 18A, in which, though DC was presumably decreasing so that AGr is presumably greater than STn, yet the AGr was only 0.54 and 0.53 per cent. at 1,008° and 1,000° respectively, or but little more than half of the lowest AGr in any lower-silicon iron the AGr of which, at 1,000°, has been preserved by quenching. In the same way the AGr in the high-silicon iron No. 18A falls to 0.19 per cent. at 800°, or only about half as much as the lowest AGr in any lower-silicon iron of which the AGr at 800°, or even at 750°, has been preserved by quenching. In short, an increase in the silicon-content is accompanied by a decrease of about half in the agraphitic-carbon content.

Striking as is the apparent influence of silicon in these cases in lowering the AGr, there is nothing to show whether its action is purely catalytic or whether part at least of its effect is due to its decreasing the solubility. And what is true of this evidence seems to me true of all the rest,[49] including some which, at first sight, seems to show directly that silicon lessens the solubility. The evidence to which these last words refer will be considered in § 44.

§ 44. *Cases in which the lowering of the agraphitic carbon by silicon may really represent nothing but the formation of cementite in graphitiferous iron. Comparison of Cases* 17 *and* 17*A with* 18 *and* 18*A*. Certain of the cast-irons of cases 17 and 18 which had been cooled slowly to the room-temperature (column 3 of Table I.) were then reheated to known temperatures and

[49] There is one piece of evidence which tends to show that the action of silicon is something more than catalytic, and hence that it really lessens the solubility. This is, that an increase in silicon-content does not seem to lower the AGr so greatly at 700° and 800° as at 1,000° and 1,100°. Now the higher the temperature the less should be the lag, and the less conspicuous, therefore, ought to be the influence of a purely spurring catalytic action in lessening that lag. Hence the apparently greater influence of silicon in lessening DC at 1,000° and 1,100° than at 700° and 800° is more readily explained by its actually decreasing the solubility than by its having a purely catalytic action. Even if this evidence is taken as only a straw to show which way the wind blows, I give it as a straw to which, perhaps, others may later be added with convincing effect.

This greater influence at the higher than at the lower temperatures is shown in the following numbers, taken, as usual, from the lowest results in each group of cases in which DC is presumably decreasing.

TABLE X.—*Silicon Lessens the Agraphitic Carbon more at High than at Low Temperatures.*

Case.	Si.	700°	800°	900°	1,000°	1,100°
		AGr. (C.C.)	AGr. (C.C.)	AGr. (C.C.)	AGr. (C.C.)	AGr. (C.C.)
3	0.80	0.38	0.43	0.99	1.29	1.26
5	2.10	0.28	0.54	0.88	0.93	1.00
Difference		+0.10	—0.11 (750°)	+0.11	+0.36	+0.26
19	0.10±		0.31			
20	0.10±				0.92	
5	2.10				0.93	1 00 (1,050°)
18A	4.23		0.19		0.53	0.56
Difference			+0.12		+0.40	+0.44

The designation numbers of these cases are the same as those used in Table I.

quenched, so as to preserve the condition then reached. After the original slow cooling to the room-temperature the AGr was naturally much less in the high- than in the low-silicon iron. But the important point is that, whereas in the reheatings to 800° and 900° the AGr of the high-silicon iron seems[50] to have decreased, and hence though low yet to be greater than the solubility for 800° and 900°, yet the AGr of the lower-silicon iron seems[50] to have increased, which (if we ignore variations in Cem. C) suggests that though high it is less than the solubility for 800° and 900°.

This seems to argue that the solubility is less for the high- than for the low-silicon iron, or, in short, that silicon lowers the solubility. But this increase of AGr in the low-silicon iron may have been due wholly to a large increase in cementite, and there may really have been a decrease in DC masked by this increase in Cem. C. If so, this case would no longer suggest that silicon lowers the solubility.

§ 45. *Formation of cementite from graphite.* That cementite may well form under these conditions is indicated by a comparison of cases 19 and 20 of Table I. The fact that the AGr of case 20 fell at 1,000° from 2.82 to 0.92 shows that ST 1,000° must be at least as low as 0.92 per cent. With this number, 0.92 per cent., in mind, the much larger numbers, 1.08 at 1,000° and 1.36 at 1,050°, to which the AGr of No. 19 increased from 0.41 per cent., cannot be regarded as representing DC. STn cannot be thought to have jumped from 0.92 per cent. at 1,000° to 1.36 per cent. at 1,050°. The silicon and other elements seem to have been alike in these cases. Hence we infer that the excess of AGr in case 19 must be due to the formation of much, indeed, at 1,050° of very much, cementite.

Thus we see that our inability to infer from the comparison of cases 17, 17A, 18, and 18A whether silicon does or does not lower the solubility is due to our uncertainty as to the quantity of cementite present. If we knew that the increase of AGr in these low-silicon irons was not due to a large increase in cementite, then we should have strong reason to think that

[50] I say "seems to" because the discrepancies between the different analyses which Heyn and Bauer with a proper frankness give in detail, may raise a doubt in some minds whether the increase and decrease of which I here speak really took place.

silicon does lower the solubility. Thus it seems to me that the first thing needed to aid our study of the effect of silicon is quantitative determination, or at least micrographic estimation, of the cementite present, a labor which I commend to our metallographers. I recognize the difficulties which the quenched condition of the specimens may cause in this estimation.

§ 46. *The position of the curve of solubility as shown by the present data.* What assumption shall we make as regards the effect of silicon on the solubility? Let us assume that silicon does not lower the solubility, but in examining the results of our study let us consider which of them rest on this assumption and which do not. To the latter class belong the results reached with irons nearly free from silicon. Indeed, it seems to me that this is all that can be done, because our data are too scanty to permit us to make a series of inferences, a first as to silicon-less irons, a second as to those with a moderate quantity of silicon, a third for those with somewhat more, and so on.

§ 47. *Solubility at* 500°. Case 14 of Table I. indicates that the solubility falls to zero somewhere above 500°. Here cast-iron containing initially 0.25 per cent. of graphite and 2.71 per cent. of agraphitic carbon, though it underwent no change on staying for 3 hr. at 500°, had the whole of its combined carbon changed to graphite when held for 3 hr. at 600° and cooled slowly. From the fact that no combined carbon changed to graphite at 500°, in spite of the initial presence of 0.25 per cent. of graphite to act as a nucleus to start graphitization, I infer that it is probable that graphitization will not occur at temperatures as low as 500°. Hence I infer that the complete change of the 2.71 per cent. of combined carbon into graphite, which occurred in the heating to 600° and cooling thence, must have occurred before the temperature had sunk as low as 500°, and hence that the solubility must have sunk to zero at some point higher than 500°.

§ 48. *Indications that the solubility may be very low at a relatively high temperature.* There are two facts which support the belief that the solubility falls nearly or quite to zero at a temperature at least moderately high. Of these, the first is the familiar one that many commercial cast-irons are practically free from agraphitic carbon, even if their cooling is not very

slow.[51] The cooling of the cast-iron in the sand mold in the casting-house of the iron blast-furnace is rarely extremely slow. In general, the pigs are broken up and thrown out of the sand soon after they have solidified, for the important practical reason that they can then be broken off easily from the sow or runner. Thus disturbed and exposed to the air, even to the cold air of winter, they are often nearly free from agraphitic (combined) carbon. This implies not only that STn has fallen nearly or quite to zero, but further that the temperature at which it has thus fallen must be high enough to enable the dissolved carbon actually to obey this low solubility and to fall thus nearly or quite completely out of solution.

The second of these two facts is that, in many micrographs of cast-iron, either large regions or the whole of the area shown consists of nothing but pure graphite and apparently pure ferrite. To me this indicates, first, that at some temperature or other, Ty, the solubility of the carbon in the iron has fallen to zero; second, that this temperature, Ty, is a rather high one, probably well above 700°, for an extremely simple reason. The actual precipitation of the carbon from the austenite as graphite in cooling must have extended at least down to Ty, and may, through lag, have extended below Ty. Now, the fact that the graphite which has thus separated has coalesced completely into large islands, leaving no visible uncoalesced islets of graphite, indicates that Ty must be high enough to give to the matrix of austenite such mobility as will permit the initially scattered islets of graphite thus to coalesce completely into large islands with considerable rapidity. That the temperature which does this is above 700° I infer from the facts that the sheetlets

[51] I find the following cases, which seem especially deserving of confidence:

	Combined (agraphitic) Carbon in Cast-Iron. Per Cent.
Percy, *Metallurgy, Iron and Steel*, p. 536, ...	0.00 (Abel).
Percy, *Metallurgy, Iron and Steel*, p. 547, ...	0.04 (E. Riley).
Ledebur, *Eisenhüttenkunde*, fifth edition, vol. ii., pp. 20, 21 (1906), ...	0.00 ferrosilicon 0.00 ferrosilicon. 0.15 No. 3 pig-iron. 0.13 No. 3 pig-iron. 0.17 No. 3 pig-iron.

Besides these, which seem to be commercial cast-irons, the very many commercial published analyses with but a very small quantity of combined carbon deserve a certain weight.

of cementite and ferrite, interstratified in steel as pearlite, do not rapidly coalesce at 700°, as is shown by the fact that well-formed pearlite, showing no sign of such coalescence, remains after even rather slow cooling from 700° down; indeed, after a cooling probably much slower than that which, in the casting-house of the iron blast-furnace, seems to suffice to permit this complete coalescence of the islets of graphite separated above Ty.

We may therefore adopt provisionally, as a working hypothesis, that the solubility of carbon in iron in presence of graphite falls to zero at some temperature certainly not below 500°, probably not below 700°.

Of great importance in this connection is the observation of Goerens and Gutowsky that, in a hypo-eutectic cast-iron which they quenched at 1,135°, the beginning of the eutectic freezing, the primaustenoid pine-trees are crossed with needles of cementite where they adjoin the cementite-austenite eutectic, but are free from them (and also it appears from graphite) where they adjoin the graphite-austenite eutectic. The freedom from cementite needles argues that these latter primaustenoid masses obeyed the graphite-austenite diagram in their very birth. But if this is admitted, then their freedom from graphite argues that, as these pine-trees grew in cooling through region 2 of Fig. 1, p. 5, no graphite was precipitated, because if any had been, it should be visible in these pine-trees. Next, if no visible quantity of graphite is precipitated, is not that an indication that there is no considerable decrease in the solubility of carbon in presence of graphite in cooling through region 2? But, if the solubility were considerable, then there ought to be a considerable decrease of it and a considerable precipitation of graphite. The absence of this precipitation, then, is a suggestion that, even at this high temperature, the solubility of carbon in presence of graphite is small.

Beyond inferring that the solubility of carbon in presence of graphite is probably less than others have believed, we may here begin to suspect that it is exceedingly small, or perhaps even *nil*, even at high temperatures. This observation of Goerens and Gutowsky gives us a hint in that direction. The absence of fine particles of graphite from some graphitic irons just noted gives us another such hint. On the other hand, our

present theory implies that the first step in the now well-established change of graphite into cementite is the dissolving of that graphite in the austenite, to be followed by reprecipitation of the carbon as cementite. But the rapidity with which this change occurs certainly suggests that this dissolving of graphite must go on rapidly, and hence that the solubility of graphite is considerable. Or is our theory wrong?

§ 49. *Solubility at* 700°. In § 43 we inferred from case 5 of Table I. that the solubility at 700° ought to be 0.28 per cent. or less. Applying expression (6) to case 19 of Table I., we find a number very near to this, 0.31 per cent. Therefore, we may infer provisionally that the solubility is not greater than 0.28, even in case the silicon-content is low. The still lower number, 0.17 in case 14, should be rejected, because it may have been reached during the slow cooling below 700°.

§ 50. *Solubility at* 800°. On the principle that the lowest trustworthy result must be taken as that which is least in excess of the true solubility, the number 0.54 per cent., based in § 43 on case 5, should be replaced by the still lower number, 0.19 per cent., based on case 18A. This is, indeed, a very high silicon iron, but the influence of its silicon is likely to have been offset in large part by its high sulphur-content. As in case of 700°, so here the still lower number, 0.12 per cent. of case 14, must be rejected, because of the opportunity for the dissolved-carbon content to decrease below ST 800° in the slow cooling from 800° down.

§ 51. *Solubility at* 900°. The very close agreement between six of the seven results in cases 4 and 5 at first sight raises a certain presumption that they represent truly the solubility at 900°. How, it may be asked, is it possible that these six results should all be out of equilibrium, either through containing dissolved carbon in excess of the true solubility or through containing cementite not yet changed into graphite, and yet should agree so closely? But we have already seen that, harmonious as these results are, they are to be rejected because contradicted by the more trustworthy results of this same case 5, 0.54 per cent. for 800° and 0.93 per cent. for 1,000°. They thus appear to be too high even for this relatively low silicon iron. So, too, we must reject the other semi-direct datum as to ST 900°—viz., the falling of AGr to 0.62 per cent. in the high-silicon iron

No. 18A, because still lower AGr is reached at 1,000° in case 18A, as the next section shows. Hence, because ST 900° is not indicated by any trustworthy results at 900°, it must be reached by interpolation between ST 800° and ST 1,000°.

§ 52. *Solubility at* 1,000°. Among the very low silicon irons there seems, at first sight, to be a cogently close agreement between the falling AGr of cases 20 and 21, 0.92 and 0.95 per cent., and the rising AGr of case 19, 1.08 per cent. But, as already explained, I incline to reject all of these summarily, the 1.08 per cent. because it probably represents much Cem. C, just as the AGr of this same case at 1,050°, 1.36 per cent., certainly represents much Cem. C, and the 0.92 and 0.95 per cent. of cases 20 and 21, because there is every reason to believe that, with this extremely low silicon, about 0.10 per cent., the lag is far too great to permit AGr to approach STn closely in the short time here given, so that they, too, are probably much greater than STn. In the 4-hr. exposures to 1,000° of cases 2 and 3, with much more silicon, the AGr evidently remained very much greater than STn, because much greater than the AGr of cases 20 and 21. The intrinsic improbability that AGr has closely approached STn in the short exposures of cases 20 and 21 is far too great to be set aside because of the agreement between these and the 1.08 per cent. reached in case 19, especially in view of the fact that the difference is wrong in sign. The falling AGr of cases 20 and 21 should remain greater than STn, the rising one of case 19 should remain less than STn, and thus, *a fortiori,* less than those of cases 20 and 21, whereas it is actually 0.16 per cent. greater than that of case 20. These results, then, would carry no great weight even on the theory that silicon does not lower the solubility of carbon, STn. All that can be said confidently for them is that, among the low-silicon results reached at 1,000°, these are the ones least in excess of ST 1,000°.

In short, on the theory that silicon does lower the solubility these results, as applicable to low-silicon iron, are questionable, because of too short exposure to 1,000°; on the other theory, that silicon does not lower the solubility, they are, of course, flatly contradicted by the much lower AGr reached, 0.54 in case 18, 0.53 in case 18A.

To sum this up, the numbers we have here selected as the

least in excess of STn, *i.e.*, as giving an upper limit, at an indeterminate distance below which STn must lie, are those of the first four lines of Table XI., the last line of which gives for comparison the numbers put forward by some previous writers in the incorrect belief that they were supported by the results of Charpy and Grenet and of Mannesmann. Whether for high or for low silicon, these latter numbers are evidently above the true upper limit of solubility, and they may be very much above the solubility itself.

TABLE XI.—*Provisional Upper Limits for the Solubility of Carbon in Austenite in the Presence of Graphite.*

(The true solubility should not be greater than these numbers and may be less than them by an indeterminate amount.)

Temperature		500° to 600°.	700°.	800°.	900°.	1,000°.
Silicon-Content.	Case No.	Upper Limit of Solubility.				
		C. Per Cent.	C. Per Cent.	C. Per Cent.	C. Per Cent.	C. Per Cent.
0.10±	20	.....	.....	.....	.....	0.92
2.10	5	.....	0.28	0.54	.....	.....
3.15	14	0	0.?	.....	.....	.....
4.23	18A	.....	.....	0.19	.....	.....
Numbers Offered by Some Previous Writers.						
.....	.....	.....	0.70	.....	0 90	1.30

These numbers must be taken as only very rough approximations to the truth, reached by means of assumptions which are open to question. Indeed, I doubt whether they are worth intrinsically what it has cost to reach them. My purpose in seeking them has been to show how great caution should be used in attempting to determine the solubility.

§ 53. *The formation of cementite in reheating graphitiferous iron.* § 45 showed that the high AGr of case 19 at 1,050° was probably due to the formation of much cementite in an iron already very graphitic, *i.e.*, to a change from graphite into cementite. This change may well have been an indirect one, the graphite first dissolving in the austenite and then re-crystallizing out as cementite. The same thing seems to have happened when

Royston (case 27) on reheating black-heart malleable cast-iron, of which the carbon was apparently wholly graphitic, made 0.85 per cent. of the carbon recombine at 720° and 1.50 per cent. at 1,030°. Both these numbers are very much above the upper limit for STn, shown in Table XI., and *a fortiori* greater than the probable DC. Their excess over DC seems to represent cementite formed from the original graphite.

§ 54. *Influence of silicon on the eutectic carbon-content.* Heyn and Bauer[52] make the interesting observation that cast-iron containing 4.16 per cent. of silicon and 3.12 per cent. of carbon is practically of eutectic composition, as is shown by the fact that the retardation in freezing is confined within very narrow temperature-limits. This goes to show that the presence of this 4.16 per cent. of silicon has shifted the eutectic ratio from its normal position in silicon-less iron, 4.30 per cent., to the neighborhood of 3.12 per cent., and to confirm my inference to the same effect, from the fact that, if the pig-iron is rich in silicon, then much kish is found blowing about the mixers in which the iron is stored, but not if the silicon is low.[53] This kish is probably the primary graphite, which rises to the surface by gravity, and so, because of its lightness and of the breadth of its flakes, is readily thrown out of the molten metal and into the air by any small quantity of escaping gas.

§ 55. *Meaning of the retardation at* 1,050°. Following Roozeboom, Dr. Benedicks explains the retardation at 1,050° by supposing that it represents the precipitation of cementite in the interior of the eutectic austenite, as, by fall of temperature, this becomes sufficiently supersaturated with cementite to induce this precipitation in the absence of nuclei of cementite.[54] At the surface of the masses of eutectic austenite, the bands of eutectic cementite, acting as nuclei, might suffice to cause the cementite to precipitate out of the eutectic austenite nearly as fast as the solubility of carbon in it decreases, or at least to prevent great lag; but in the interior of these masses, where such nucleus-action is lacking, a considerable degree of supersaturation may occur, and, once the excess of cementite then

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[52] *Stahl und Eisen*, vol. xxvii., No. 44, p. 1566 (Oct. 30, 1907).
[53] *Iron, Steel, and Other Alloys*, 2d edition, p. 210 (1906).
[54] *Metallurgie*, vol. iii., No. 13, p. 434 (July 8, 1906).

begins to separate, its separation may be so rapid as to retard the cooling. This temperature, 1,050°, then would in a way correspond to a change from a metastable to a labile condition, from a condition in which an overdue transformation occurs only in case it is stimulated by the presence of nuclei of the stable substance to one in which it occurs even in the absence of such nuclei.[55]

However well this may explain the occurrence of this 1,050° retardation in cooling from the molten state, it does not seem to explain easily its occurrence, which Dr. Offerhaus and I have repeatedly noticed, in the cooling of pieces reheated apparently to well below incipient fusion. In these pieces it is hard to believe that there should not remain enough undecomposed cementite within the austenite to give the nucleus action needed to start the re-precipitation of cementite as the solvent power of the austenite decreases.

Appendix.

Diagram indicating the relation between the carbon-content and the proportion of each of the constituents present, *Fig.* 10. This diagram is on the general plan so well used by Prof. Sauveur. Abscissæ indicate the carbon-content, the ordinates of the lines AB and BN indicate the quantity of pearlite, assuming that it retains the whole of its initial mass, and that none of either of its constituents, ferrite or cementite, has coalesced, and so ceased to retain the original pearlitic structure. The proportion of pro-eutectoid ferrite is given (as in Prof. Sauveur's original diagram) by the intercept ED, the primary cementite by MN, the eutectic cementite by KJ, and the pro-eutectoid cementite by JF.

The general procedure in calculating the quantities of primary, eutectic, pro-eutectoid, and eutectoid cementite is as follows:

Case 1, *carbon* 4.30 *per cent.* The whole mass at 1,130° consists of a eutectic of austenite and cementite. The austenite contains 2.20 per cent. of carbon, the cementite 6.67 per cent. of carbon. The carbon in the x per cent. of austenite plus the

[55] See interesting determinations of the passage from the metastable to the labile state in many liquids, water included, Miers and Isaac, *Transactions of the Chemical Society*, vol. lxxxix., p. 413 (1906), and *Chemical News*, vol. xciv., No. 2439, p. 89 (Aug. 24, 1906); also Isaac, *Transactions of the Chemical Society*, vol. xciii., p. 384 (1908).

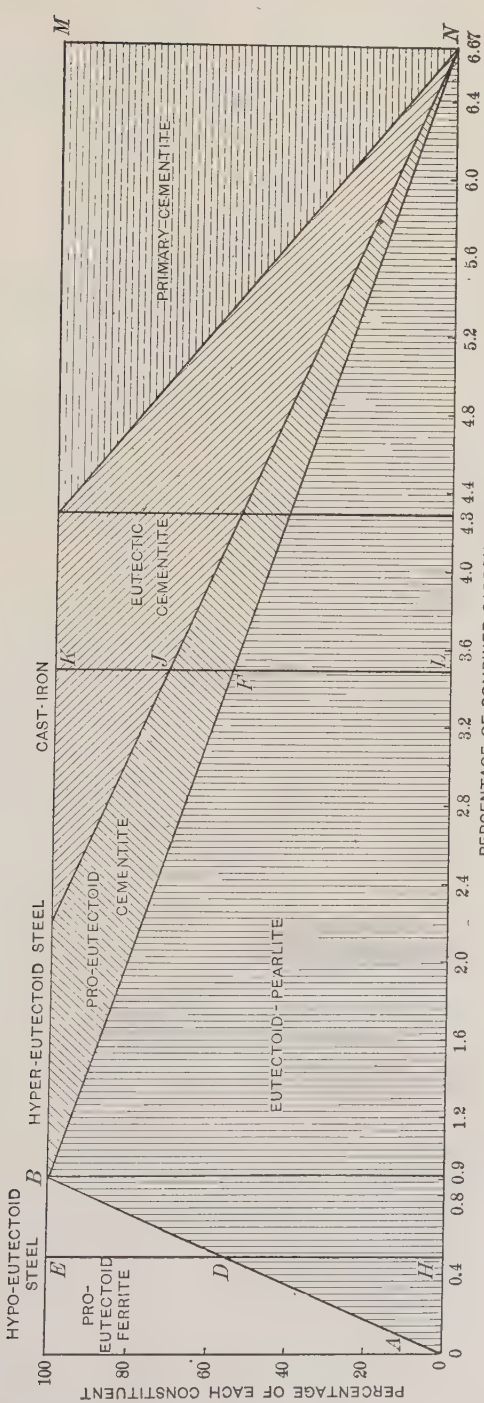


FIG. 10.—RELATION BETWEEN THE CARBON-CONTENT AND THE PERCENTAGE OF THE SEVERAL CONSTITUENTS OF SLOWLY-COOLED STEEL AND WHITE CAST-IRON.

y per cent. of cementite is, by assumption, 4.30 per cent. of the whole mass.

Let x = the percentage of the austenite in the whole mass at 1,130°.

Let y = the percentage of the cementite in the whole mass at 1,130°.

(1) $0.022\ x + 0.0667\ y = 4.30$.

(2) $x + y = 100$.

Solve for y, the eutectic cementite, which persists unaltered in the cold.

Case 2, *carbon* 4.50 *per cent.* At 1,130° the mass consists of the eutectic of austenite of 2.20 per cent. carbon and cementite of 6.67 per cent. carbon, plus the primary cementite, also of 6.67 per cent. carbon.

Let x = the percentage of the eutectic austenite in the whole mass at 1,130°.

Let y = the percentage of the eutectic cementite in the whole mass at 1,130°.

Let z = the percentage of the primary cementite in the whole mass at 1,130°.

The carbon in the x per cent. of 2.2 per cent. austenite plus that in the y and z per cent. of eutectic and primary cementite, by assumption, is 4.5 per cent. of the whole mass.

(3) $0.022\ x + 0.0667\ (y + z) = 4.50$.

The ratio $x : y$ is here the same as in case 1; hence,

(4) $x : y = 53 : 47$.

(5) $x + y + z = 100$.

Solve for x, y, and z.

In cooling through region 7 the primary and the eutectic cementite persist unchanged, but the eutectic austenite, initially of 2.20 per cent. of carbon, is gradually impoverished in carbon by giving birth to the pro-eutectoid cementite, until at Ar_1 it becomes hardenite of 0.90 per cent. of carbon.

Let u = the hardenite present at Ar_1, and

Let v = the pro-eutectoid cementite there present.

The $0.022\ x$ carbon which at 1,130° was present in the austenite will now be distributed between the hardenite and the pro-eutectoid cementite. Hence,

(6) $0.009\ u + 0.0667\ v = 0.022\ x$.

(7) $u + v = x$.

Solve for u and v.

NOTES TO TABLE I. (pp. 68, 69).

Nos. 1 to 5. Charpy and Grenet, *Bulletin de la Société d'Encouragement pour l'Industrie nationale*, vol. cii., p. 399 (1e semestre, 1902). Five cast-irons, of the composition given below, were first melted and granulated in cold water, after which they contained no appreciable proportion of graphite, except No. 5, which had 0.20 per cent. After exposure to the temperatures indicated they were quenched in cold water.

a In these cases the metal was cooled at the rate of 50° per hr. from 1,170° to 900°, and then allowed to cool completely over night. The next day the specimens were reheated to 900°, and then again cooled at the rate of 50° per hr. to the temperatures here given, held there for 2 hr., and then quenched in cold water.

b In these cases the cast-iron was heated directly to the temperature given in this table, and after staying there was quenched in cold water.

c In these cases the metal was first raised to between 1,100° and 1,170°, then cooled slowly to the temperatures given in this table, and after staying there for the time indicated was quenched in cold water.

Chemical Composition of Irons, Nos. 1 to 5.

	Carbon. Per Cent.	Manganese. Per Cent.	Sulphur. Per Cent.	Phosphorus. Per Cent.
No. 1, . . .	3.60	0.03	0.01	traces
No. 2, . . .	3.40	traces	0.02	0.02
No. 3, . . .	3.25	traces	0.02	0.03
No. 4, . . .	3.20	0.12	0.01	0.01
No. 5, . . .	3.30 { C. C., 3.10 / Gr., 0.20 }	0.12	0.02	0.01

Nos. 6 to 14. Wüst and Schlösser, *Stahl und Eisen*, vol. xxiv., No. 19, pp 1120–3 (Oct. 1, 1904).

No. 16. Benedicks, *Metallurgie*, vol. iii., No. 13, p. 435, bottom (July 8, 1906).

Nos. 17 and 18. Heyn and Bauer, *Stahl und Eisen*, vol. xxvii., No. 44, pp. 1565–71 (Oct. 30, 1907); No. 45, pp. 1621–5 (Nov. 6, 1907). They melted lots of 400 g. each of cast-iron, made by mixing together white cast-iron and ferrosilicon. The composition of these, and of the resultant cast-irons, was as follows:

	Carbon.			Si. Per Cent.	Mn. Per Cent.	P. Per Cent.	S. Per Cent.	Fe. Per Cent.
	C. C. Per Cent.	Gr. Per Cent	Total. Per Cent.					
Initial white iron	2.88	0.07	2.95	0.63	0.22	0.09	0.33	.....
Initial ferrosilicon	.....	.....	.....	96.	0.27	0.041	0.108	3.50
Resultant cast-iron, No. 17:								
Maximum	.....	.....	3.33	1.72	.....	.....	.....	.....
Minimum	.....	.....	3.05	1.20	.....	.....	.....	.....
Average	.....	.....	3.19	1.47	.....	.....	.....	.....
Resultant cast-iron, No. 18:								
Maximum	.....	.....	3.14	4.27	.....	.....	.....	.....
Minimum	.....	.....	2.59	2.90	.....	.....	.....	.....
Average	.....	.....	2.93	3.89	.....	.....	.....	.....

The sulphur-content was regrettably high, so high, indeed, that the results are not closely comparable with those of other experimenters who have treated cast-iron of a more normal composition. Each lot of cast-iron was melted in a

TABLE I.—*Agraphitic (Total Combined) Carbon of Cast-Iron After Long Exposure to Certain High Temperatures.*

No.		Silicon Content. Per Ct	Room T° AGr. (C. C.) P.ct	Hr	500° AGr. (C. C.) P.ct	Hr	600° AGr. (C. C.) Per Cent.	Hr	700° AGr. (C. C.) Per Cent.	Hr	800° AGr. (C. C.) Per Cent.	Hr	900° AGr. (C. C.) Per Cent.	Hr	1,000° AGr. (C. C.) Per Cent.	Hr	1,050° AGr. (C. C.) Per Cent.	Hr	1,080° AGr. (C. C.) Per Cent.	Hr	1,100° AGr. (C. C.) Per Cent.	Hr	1,150° AGr. (C. C.) Per Cent.	Hr	1,170° AGr. (C. C.) Per Cent.	Hr	1,200° AGr. (C. C.) Per Cent.	Hr
1	Charpy and Grenet.....	0.07	.....	...	.....	...	.....	.....	0.43*a*	2	1.31*a*	2	.....	.....	1.74*c*	2	.....	.....	.....	.....	1.74*c*	2						
2		0.27	.....	...	.....	...	.....	.....	0.43*c*	4	.....	.....	.....	.....	1.74*c*	4	.....	.....	.....	.....	1.74*c*	4	.....	.....	2.61*b*	2		
			.....	...	.....	...	.....	.....	.....	.....	.....	.....	.....	.....	1.62*c*	4	.....	.....	.....	.....	1.93*c*	4						
3		0.80	.....	...	.....	...	.....	.....	0.38*a*	2	3.19	1	2.97	1	2.94	1	.....	.....	.....	.....	1.48*c*	2						
			.....	...	.....	...	.....	.....	0.38	4	0.43*a*	2	2.40	2	1.41	2												
			.....	...	.....	...	.....	.....	.....	.....	.....	.....	0.99*c*	2	1.55*c*	2	.....	.....	.....	.....	1.26	4	.....	.....	1.69*b*	2		
			.....	...	.....	...	.....	.....	.....	.....	3.07	4	1.14	4	1.29	4												
			.....	...	.....	...	.....	.....	.....	.....	.....	.....	1.17	4	1.52	4												
4		1.25	.....	...	.....	...	.....	.....	3.42	1	3.08	1	0.90	1														
			.....	...	.....	...	.....	.....	3.30	2	2.47	2	0.90	2														
			.....	...	.....	...	.....	.....	3.13	4	1.56	4	0.99	4	0.98	4	.....	.....	.....	.....	1.02	4						
5			.....	...	.....	...	.....	.....	.....	.....	.....	.....	0.90	4														
		2.10	.....	...	.....	...	650°	1	1.90	1	0.78	1	0.88	1														
			.....	...	0.47+	6	1.19	2	0.89	2	0.90	2	0.93	4							1.00	4						
			.....	...	.....	...	0.28	4	0.54	4	0.90	4																
6	Wüst and Schlösser.....	0.05	2.11	...	.....	...	.....	.....	.....	.....	.....	.....	.....	.....	1.98	3	.....	.....	.....	.....	1.47	3	.....	.....	.....	.....	1.43	3
7		0.078	3.53	...	.....	...	.....	.....	.....	.....	.....	.....	3.50	3	3.47	3	.....	.....	.....	.....	2.70	3	1.89	3	.....	.....		
8		0.13	4.36	...	.....	...	.....	.....	.....	.....	.....	.....	4.24	3	3.93	3	2.10	3	.....	.....	1.86	3						
9		0.26	3.29	...	.....	...	.....	.....	.....	.....	.....	.....	3.27	3	3.02	3	.....	.....	2.77	3								
10		0.55	3.06	...	.....	...	.....	.....	.....	.....	.....	.....	2.73	3	1.98	3	.....	.....	0.69	3								
11		0.81	2.70	...	.....	...	.....	.....	.....	.....	2.47	3	0.73	3	0.70	3	.....	.....	.....	.....	0.49	3						
12		1.20	2.44	...	.....	...	.....	.....	.....	.....	2.16	3	0.62	3	0.57	3	.....	.....	.....	.....	0.55	3						
13		2.12	2.21	...	.....	...	1.67	3	0.34	3	0.32	3	0.29	3	0.35	3	.....	.....	.....	.....	0.37	3						
14		3.15	2.71	...	2.71	3	0.00	3	0.17	3	0.12	3	0.10	3	0.12	3	.....	.....	.....	.....	0.14	3						
16	Benedicks, solubility represented by r, r′ and r″.....	.....	.....	...	.....	...	.....	.....	.....	.....	0.70 r	.....	0.90 r′	.....	1.30 r″	.....												

17 — Heyn and Bauer.....

A. Combined + Dissolved Carbon left after a Slow Cooling to these several temperatures, interrupted by Quenching at these temperatures.

1.20 to 1.72	1.25 to 1.32	Room	.....	...	.....	.....	.....	1.19	T° 708°	.....	.....	1.21	T° 907°	.....	.....	1.76 1.49	T° 1,057° 1,038°	1.79 1.64	T° 1,087° 1,085°	2.80	1,108°	2.99	1,159°	.....	.....	2.97	T° 1,360°
																										3.02	1,256°

B. On Reheating Slowly-Cooled pieces of No. 17 to these several temperatures, holding for ½ hour, and then Quenching, the Agraphitic (Combined) Carbon increased from between 1.16 and 1.51 to

17A			.....	...	.....	...	.....	.....	.....	.....	1.50	.....	1.51	.....													1.67	½ hr. T°
			.....	...	.....	...	.....	.....	.....	.....	1.54	½	1.43	½														
		1.46	.....	...	.....	...	.....	.....	.....	.....	.....	.....	.....	.....													1.73	1,180° to 1,200°

| | | |

graphite crucible under a layer of charcoal, and then allowed to cool slowly in the furnace itself, either to the room-temperature, or to a certain specified temperature, QT, on reaching which the iron was quenched in water, so as to catch as nearly as practicable the condition reached in the slow cooling. The rate of cooling seems to have been about as follows:

	From 1,200° to 1,100°.	1,100° to 1,000°.	1,000° to 900°.	900° to 800°.	800° to 700°.	700° to 600°.
Minutes, estimated, 6	9	12	15	18	26	

Certain of the slowly-cooled pieces were then reheated to the temperatures given under 17A and 18A, and then held there for either 0.5 hr. or 2 hr., and then quenched in water.

No. 19. Charpy, *Revue de Métallurgie*, vol. v., No. 2, Mémoires, p. 77 (Feb., 1908). Swedish wrought-iron was cemented and melted in a crucible with an excess of wood-charcoal, and allowed to cool slowly in the crucible, when it contained 3.34 per cent. of graphite and 0.41 per cent. of combined carbon, together with "about 0.10 per cent. of silicon and of manganese, and traces of sulphur and phosphorus." Pieces of this metal 4 cm. cube were then held at the temperatures stated in the table for from 6 to 7 hr. in the 750° and 850° heatings, and 3 hr. in the others, then quenched in water, and reheated to 500° to soften them enough to be cut. 2 mm. were then cut off the faces of each cube. Then filings were taken for analysis.

No. 20. *Idem.* He proceeded as in No. 19, except that the molten metal was cast in a metallic mold, when it contained 2.82 per cent. of combined carbon and no graphite. It was then reheated to 1,150°, doubtless to start the formation of graphite, then cooled very slowly to 1,000° and quenched. After rémoving the oxidized outside the remainder contained 1.82 per cent. of graphite and 0.92 per cent. of combined carbon.

No. 21. *Idem.* He proceeded in the same way, except that the crucible containing the molten cast-iron was set in a furnace at 1,000°, so that the iron passed directly but very slowly to 1,000°. After staying at 1000° for 3 hr. it was quenched in cold water, when it contained 2.15 per cent. of graphite and 0.95 per cent. of combined carbon.

Here and elsewhere he reports the combined carbon as "dissolved carbon," but it seems clear that under this he includes both the dissolved and the cementite carbon.

No. 22. Royston, case B, § 16, p. 25; *Journal of the Iron and Steel Institute*, vol. li. (1897, No. I.), p. 166.

No. 23. Charpy, case H, § 16, p. 26; *Revue de Métallurgie*, vol. v., No. 2, Mémoires, p. 78 (Feb., 1908). Iron was cemented in wood-charcoal at 1,000°. The length of time is not given, and there is no indication that the metal was quenched instead of being cooled slowly. I infer that it probably was not quenched, from the fact that in many of the experiments described in the same paper he specifically asserts that the metal was quenched, and that in the experiments which he groups together with this No. 23 he omits all reference to the rate of cooling. Hence I have marked it SC (slowly cooled).

The combined-carbon content may be above the true solubility at 1,000°, because much of that carbon may have existed as cementite, as is the case in most cementations. On the other hand, it is conceivable that the combined-carbon content may be below the true solubility, because, if it should have so happened that, at 1,000°, all the cementite had been broken up and its carbon changed into graphite and dissolved carbon, and if this dissolved carbon had been just that which saturated the austenite at 1,000°, then in cooling slowly from 1,000° down, part of this

dissolved carbon would naturally have separated out as graphite because of the decreasing solubility of carbon with falling temperature. Hence the case offers no strong evidence.

No. 24. Charpy, *Comptes rendus*, vol. cxxxvi., No. 17, p. 1002 (Apr. 27, 1903), and *Iron and Steel Magazine*, vol. viii., No. 4, p. 306 (Oct., 1904). Steel filings containing 2 per cent. of carbon were heated to 1,000° for 36 hr. in pure carbonic oxide, after which they contained 9.27 per cent. of total carbon and 1.00 per cent. of combined carbon. It is probable that they were slowly cooled (SC) after this heating.

No. 25. *Idem*. A piece of soft steel of 0.09 per cent. carbon, 3 mm. in diameter, was held for 64 hr. in illuminating gas at 1,000°, and then probably cooled slowly. It then contained 7.66 per cent. of graphite and 0.66 per cent. of combined carbon.

No. 26. Sauveur, *Journal of the Iron and Steel Institute*, vol. lxxii. (1906, No. IV.), p. 506.

No. 27. Royston, *Journal of the Iron and Steel Institute*, vol. li. (1897, No. I.), p. 166.

No. 28. Goerens and Gutowsky, *Metallurgie*, vol. v., No. 5, p. 137 (Mar. 8, 1908). Pieces, each weighing about 2 g., of white Swedish cast-iron, containing C, 3.91 ; Si, 0.12 ; Mn, 0.18 ; P, 0.02, and S, 0.008 per cent., were heated separately in a magnesia-lined porcelain boat, in an atmosphere of pure nitrogen, to 1,250°, held there 4 min., then cooled slowly to the temperatures indicated in this table, and then quenched in ice-water.

The Mechanical Preparation of Ores in Sardinia.

BY ERMINIO FERRARIS, MONTEPONI, SARDINIA, ITALY.

(New York Meeting, February, 1908.)

I. Historical Review and Introduction.

The development of the mining industry in Sardinia dates from the application of the mining law of 1859, which, following the example of the French mining law of 1810, declared prospecting to be free, and suppressed the liens which had previously attached the mine to surface ownership.

The first ore-dressing plants were installed by German engineers upon the classic models of the Harz and of Freiberg; but in 1880 Sardinia commenced to open a new path in that industry, utilizing the refuse from the mining of rich, selected minerals, of poor deposits, and of mixed ores.

The first experiments were attempts to adapt the existing apparatus to local conditions, but progress was rapid, as was demonstrated already in 1887 by the calamine-dressing mill of Monteponi, which exhibited a large plant designed in accordance with the experience gained, and constructed entirely in the machine-shops of the country, after new models and designs.[1] This example was followed two years later by the Société Malfidano, with a large plant of the same sort, and with the reconstruction of the old-fashioned dressing-mill, built in 1880 by the Austro-Belgian Company. In 1900, almost all the mines of Sardinia had adopted the new designs, and the island contained 32 concentrating-works, employing 2,000 h.p. and 1,924 workmen. Since that time, there have been added a large dressing-mill for blende at Gennamari, and two for calamine, at San Giovanni and at Masua, while the establishments at other mines have been enlarged and perfected. For the treatment of intimately mixed ores, a large establishment has been built upon the new

[1] La Laveria calamine della Miniera di Monteponi, (Sardegna). By Ing. Erminio Ferraris. *Annale degli Ingegneri e Architetti italiani*, Anno IV, Fascicolo IV. (Roma, 1889).

principles at Rosas, which has solved the question of the utilization of these ores, previously neglected for lack of the means of separating them.

In considering the general case of mechanical preparation it is necessary to distinguish the two following operations: (1) the concentration of valuable minerals contained in the total product, removing the barren portions as far as possible; and (2) the separation of valuable minerals from each other, and the further removal of barren matter intimately mixed with the useful minerals.

The work of concentration of the original crude product is performed by utilizing the difference in specific gravity between the useful and the barren minerals. The usual gangue-minerals of the ores in Sardinia, dolomite, limestone, schist, clay, quartz, and diabase, which have a specific gravity not exceeding 3, are easily removed, leaving concentrates of galena, blende, smithsonite, calamine, etc., together with mixed miscellaneous products resulting from the presence of heavy minerals, such as barite, limonite, and pyrite.

Concentration is effected in two different operations: the classification of the material according to the size of the grains; and the separation of the classified grains according to their specific gravity.

II. Classification.

The two forms of classification are: (1) screening, which is used for grains generally larger than 2 mm., but can be used on 0.5-mm. grains; and (2) sorting by a current of water, which is used for grains smaller than 2 mm., fine sands and slimes.

1. *Screening.*

Until the end of 1894, screening was almost universally accomplished by the rotary trommel, but its defects, well known to all mining engineers, made a less cumbersome and more durable apparatus desirable. After many experiments, trommels were superseded with great advantage by vibrating-screens, the idea being borrowed from the modern flour-mill. At the beginning of 1898, screening at the calamine-dressing mill of Monteponi was completely transformed according to the new system of vibrating-screens, and they were at once adopted by nearly all the mills of Sardinia and Tunis.

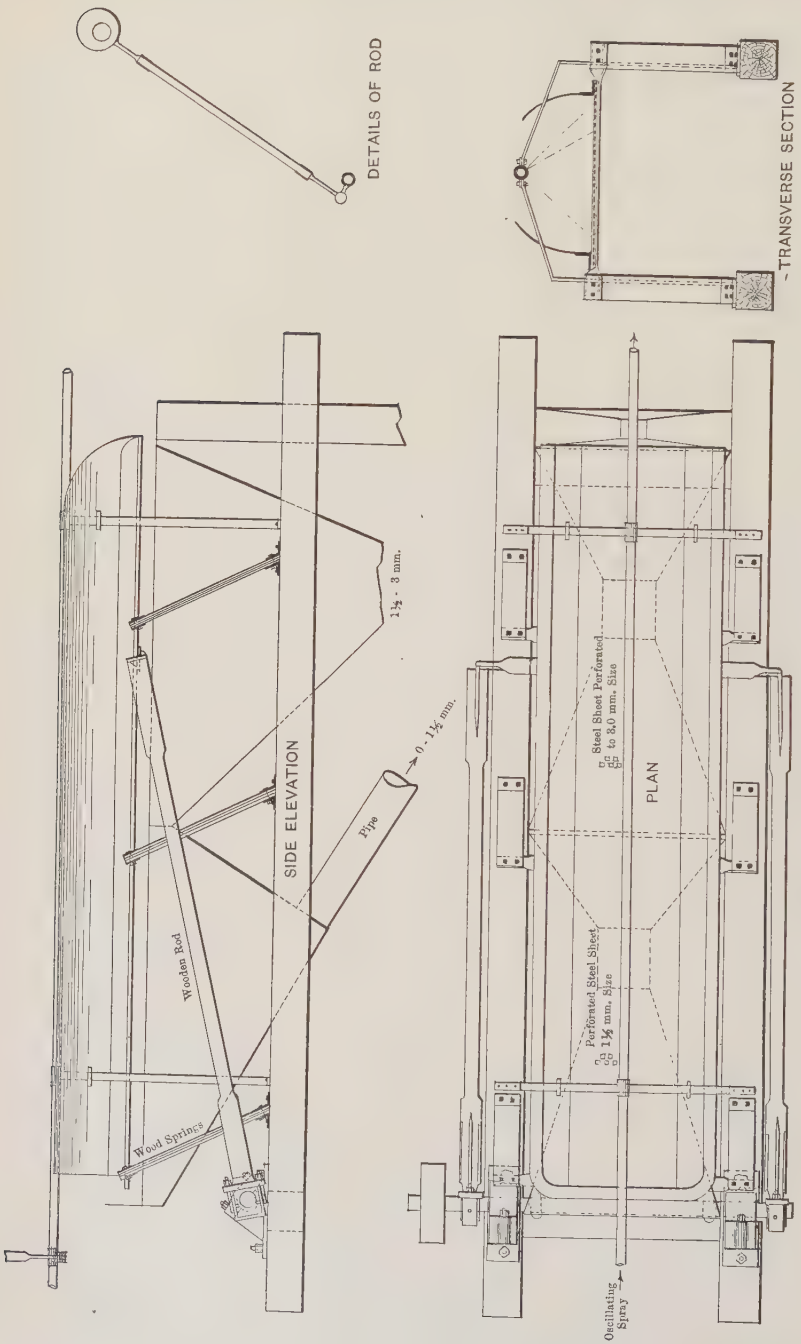


FIG. 1.—THE FERRARIS VIBRATING-SCREEN.

The Ferraris Vibrating-Screen—These screens are narrow bands of perforated sheet-iron joined by two angle-irons covering the two longitudinal edges. The width of the screen varies from 400 to 800 mm.; the length is indefinite. Fig. 1, showing the screen used for fine sizing, may be regarded as typical of the apparatus. These screens, generally with several sections perforated for different sizes, are mounted horizontally on springs made of several blade-shaped pieces of wood, in length between 400 and 600 mm., inclined about 70° from the horizontal in such a manner as to raise the screen at each oscillation in the direction of the progress of the material. An eccentric or two, and one or two wooden cranks, give to the screen from 300 to 400 oscillations per min., with an amplitude of from 30 to 40 mm. It is sprayed by a longitudinal pipe, which swings on its axis so as to sprinkle the entire width of the screen. In a complete sizing-system, the first screen, which should also cleanse off the mud, is furnished with superposed traverses to retard the course of the material, and with sprinklers arranged transversely, having jets of water pointed in a contrary direction to the material.[2]

Table I. shows an example of sizing at one of the mills at Monteponi. Screen No. 1 is fed from a hopper closed by an oscillating shovel-slide, which regulates the flow of the material on the screen.

The principal features of the three screens are summarized in Table I.

Screens Nos. 1 and 2 are supported on eight springs, each made of three leaves of beech-wood, 400 mm. in length, 80 mm. wide and 6 mm. thick. Screen No. 3 is suspended on six springs in a box in which the water-level may be regulated. This arrangement serves to clear the obstructions from the holes of the sieve by making it strike the surface of the water occasionally.

The screens may be inclined instead of horizontal, but the inclination must be contrary to the course of the material. In an installation at Monteponi the inclination was brought up to 10 per cent., in order to retard the fall of the material. The

[2] Fortschritte der Erzaufbereitung in Sardinien, *Oesterreichische Zeitschrift für Berg- und Hüttenwesen*, vol. xlviii., No. 18, p. 229 (May 5, 1900); *Ore-Dressing*, by Robert H. Richards, vol. i., p. 342 (1903).

TABLE I.—*Sizing at a Monteponi Mill.*

	Screen No. 1.				Screen No. 2.				Screen No. 3.			
Revolutions of the eccentric per minute				350				355				360
Amplitude of the oscillations, mm				34				32				30
Power employed, h.p.				3				2				1.5
Quantity of water for sprinkling per minute, liters				120				60				60
Width of the screen, mm.				600				600				600
Maximum quantity of material screened per hour, kg				30,000				8,000				4,000
Diameter of holes in sieves, mm	14	20	30	oversize	5	7	10	oversize	2	3.5	5	
Classes produced	0–14	14–20	20–30	over 30	0–5	5–7	7–10	10–14	0–2	2–3.5	3.5–5	
Length of the perforated section, m	2.2	1.5	1.2		1.8	1.5	1.2		1.20	0.80		
Average weight of material classified per hour, kg	8,800	1,800	2,000	6,000	2,800	1,200	1,200	800	800	1,200	800	

Class 0–14 goes to feed Screen No. 2. Class 0–5 goes to feed Screen No. 3.

sizing thus gains in precision, but the necessary motive-power also increases with the slope.

The Sanna Screen.—In some mills, for instance, at Malfidano,[3] sizing is pushed to 0.5 mm.; wire-gauze is used instead of perforated sheet-iron, and a long piece of angle-iron is laid on the middle of the screen, on which, if necessary, hammering is done, to remove obstructions from the meshes. With this exception the construction corresponds practically with Fig. 1.

Advantages.—The advantages of vibrating-screens over trommels in screening are as follows:

1. The wear is reduced to a minimum, either in consequence of the smaller surface necessary to screen the same quantity of material, or on account of the motion in small jumps which prevents friction on the surface of the screen.

2. The smaller height of the apparatus, which saves cost in construction.

3. A more exact sizing.

4. By using this system of sizing in coal-washeries, breaking the pieces by impact or friction is avoided. The same system may be used to drain off the sized products and to transport them horizontally.

5. Inspection and repair are simpler than with the trommels.

[3] *Rosoconti della riunione della associazione mineraria sarda.* Seduta 18 Novembre, 1900. Sanna. Nota su di uno nuova disposizione dei vagli oscillanti per la classificazione delle sabbie fini.

2. *Classification by Current of Water.*

The methods of mechanical preparation used in countries favored with an abundance of water have given rise almost everywhere to the custom of wasting water in the mills, without considering the inconveniences resulting from too great dilution of the stream which carries the fine sands. In Sardinia, the necessity for recovering the water of the washeries, in order to use it again after clarification, has taught the benefit of reducing such dilution to a minimum, and thus removing the hindrance caused by the conduits and the *spitzkasten* to subsequent concentration, while increasing at the same time, by reason of such smaller dilution, the precision of the classification itself.

In sizing by screens, the water used on the first section of the first screen is brought by pipe, together with the grains smaller than 14 mm., to the second screen, from which the water takes the sands below 5 mm., and carries them to the third screen, where the water is collected in the first hopper placed below the screen, with the fine sands below 2 mm. The mixture of water and sands flows through a pipe, carrying the sized sands to the separating-machines.

The most favorable proportion for good sorting by current of water is one portion of solid substance to five of water, but one to ten may be reached without inconvenience. Beyond that proportion, classification becomes more difficult and less exact.

The pipe carrying the sands is generally uniform in section for the entire length, while the inclination should diminish as the sands are delivered to the finishing-machines. Generally the slope begins at 30° from the horizontal (50 per cent.), and is reduced in a curve to about 4 per cent. at the first distributing-machine. The following sections have decreasing inclinations, until the last is horizontal.

The distributing-machines for classified material, called hydraulic classifiers,[4] are mounted on the pipe at the points nearest to the separating-apparatus which they feed. The distance

[4] German Patent No. 31,427; *Zeitschrift für das Berg-, Hütten- und Salinen-Wesen*, vol. xxxiv., p. 42 (1886); *Oesterreiche Zeitschrift für Berg- und Hüttenwesen*, vol. xlii., No. 35, p. 421 (Sept. 1, 1894); *Ore-Dressing*, by R. H. Richards, vol. i., p. 478 (1903).

between two hydraulic classifiers should not be less than 3 m., to give time for the material to separate during the flow.

Classification is made in the pipe by gravity and flow; the heavier grains sinking to the bottom of the pipe, while the lighter grains are carried along with a velocity which in the upper layers approaches that of muddy water. In order to secure good classification, a current of about 0.60 m. per sec. is necessary. A pipe of 100-mm. interior diameter corresponds to 100 liters of water and sand per minute. A total fall of 2 m. is sufficient for a horizontal length of 20 meters.

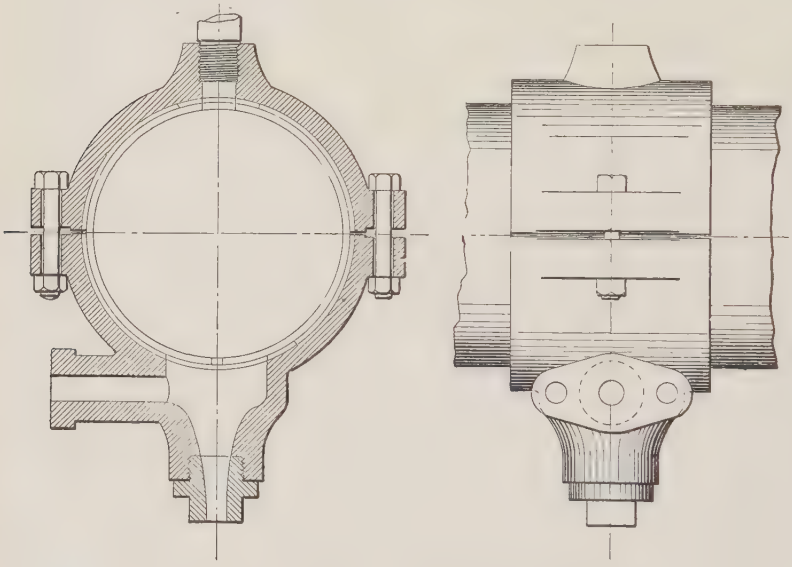


FIG. 2.—DETAILS OF HYDRAULIC CLASSIFIER.

At the point where the classified material should be discharged, a hole of from 10 to 15 mm. is drilled in the lower side of the pipe, and the conduit is surrounded by the hydraulic classifier as in a muff, as shown in Fig. 2.

The sand falls into a lower pocket, which clear water enters from the side, under pressure regulated by a valve, so as to equal the pressure of the water from the principal pipe. The sands flow away with the clear water through a lower opening from 10 to 16 mm. in diameter.

Classification is accomplished more by density than by weight or size; in fact, the heavy ores in a well-regulated con-

duit are almost entirely arrested at the first hydraulic classifiers. In treating a mixture of quartz and galena, the first of these machines delivers nearly 80 per cent. of the galena.

In treating very rich ores, pains are taken to remove all trace of sand from the muddy water before sending it to the settling-ponds. For this purpose, the conduit is lengthened horizontally, and instead of using hydraulic classifiers to draw off the grains which drag on the bottom of the pipe, use is made solely of the outlet-holes, and sands with very little water are obtained from the pipe.

The advantages of this system are evident. There is no obstruction in the works, for the pipes of the hydraulic classifiers are suspended above the separating-machines, to which they are joined by flexible rubber pipes of about 20 mm. diameter. If it is desired to discontinue the use of a separator, it is only necessary to close the orifice of the hydraulic classifier, leaving the clear-water valve a little open, in order that the hydraulic classifier may be always ready to resume its work.

The hydraulic classifier applied to the washing of coal serves to separate the barren material and the portion richest in ash, and to send to the settling-ponds the mud containing the finest particles of coal, which may be briquetted or delivered to coke-ovens.

III. Separation.

1. *Hand-Picking.*

In Sardinia, the grains more than 30 mm. in diameter are picked by hand on an endless wire belt. All the valuable ore is removed, waste being allowed to fall into cars at one end. The belt is supported by two series of rollers. At the extremities it passes around two drums, of which one provides the tension and the other the motive-power. The belt is 0.60 m. wide, woven of galvanized wire 3.5 mm. in diameter, twisted in flattened spirals, and joined by transverse wires. It is therefore easy to lay open the belt in order to shorten it, or to change a defective part.

The velocity of the belt at Monteponi is 12 cm. per sec. With favorable ores 20 cm. can be used. The height of the belt from the ground varies from 0.60 m. to 0.75 m., according to the stature of the pickers. For a length of 10 m., 1.25 h.p.

is required. If the feed is very regular, 3 tons of material may be sorted per hour. At Monteponi an average of 1,800 kg. per hour is obtained in picking material of highly irregular size.

The same system of sorting is applied at Monteponi to the tailings from jigs which treat sizes from 20 to 30 mm., and from 14 to 20 mm., in order to extract from this waste material the hydrozincite and spongy zinkiferous limonites, which cannot be separated by gravity from the dolomite. The tailings fall on one oscillating-carrier, like the vibrating-screens, but longer and not so wide, and are carried to a picking-belt 8 m. long.

2. *Hydraulic Jigs.*

Concentration of grains from 10 to 30 mm. is effected by hydraulic jigs with two compartments, and in the case of the smaller grains down to 2 mm. by jigs with five compartments.

The construction of the jigs is the same in both cases. Fig. 3 gives the details of a jig with two compartments; it is formed of three cast-iron plates which support the bearings of the eccentric shaft, joined by a wooden casing or wooden walls so as to form two communicating chambers for pistons and screens. This construction has no special advantage beyond facilitating the transportation and mounting of the jigs. But in some details the Monteponi jig differs greatly from those in general use.

The eccentrics have a variable stroke. A first eccentric fixed to the shaft is surrounded by a moving eccentric; the first has a flange which partly covers the second at the side, and both have holes through which the bolt is passed to hold them together. The holes being at a different distance in the two eccentrics, the combination forms a kind of vernier caliper, which allows variation in the eccentricity.

Eccentrics of three sizes are used: one for strokes up to 20 mm.; a second for strokes between 10 and 40 mm., and a third for strokes between 30 and 80 mm.

With five holes in each partial eccentric, 25 combinations of different strokes can be obtained between the two extremes. The superiority of the system consists in the facility with which the eccentricity can be regulated, and in the assurance that this eccentricity cannot vary during the work of the jig. This eccentric is shown in Fig. 4.

The discharge of the concentrated material is made by pipe for the coarser grains; by pipe and suction through the sieve into the hutch beneath[5] for the sands. The pipe varies in diameter from 13 to 51 mm., according to the classes treated. It

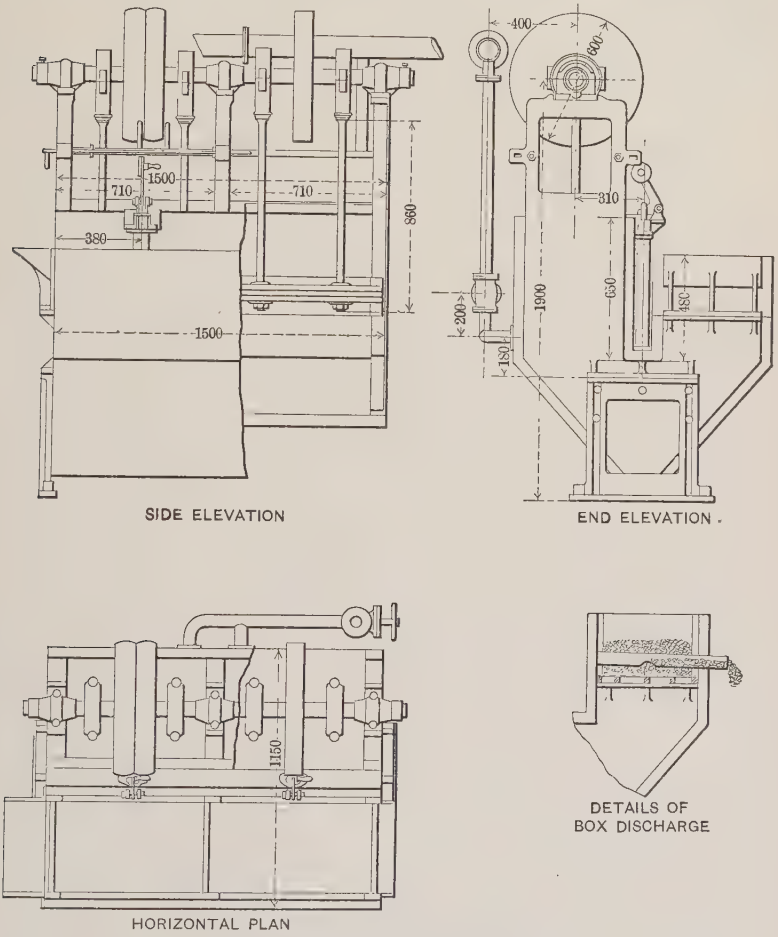


FIG. 3.—TWO-COMPARTMENT JIG.

is placed, slightly inclined towards the outside, and transversely to the screen, at about half the height of the layer of grains. On the bottom of the pipe, in the middle of the screen, a hole

[5] *La préparation mécanique des minerais métallique en Sardaigne.* Rapport présenté par M. N. Pellati, Inspecteur général des Mines, au congrés de mines et metallurgie de l'Exposition universelle de Paris en 1900.

is bored, through which the grains with the water rise through the pipe and flow away.

The jig separates the grains in layers of different density. The pipe gives an outlet to the layer of valuable mineral as fast as it rises on the screen. The discharge is made at intervals, especially for the small grains, and is stopped when waste is found mixed with the ores.

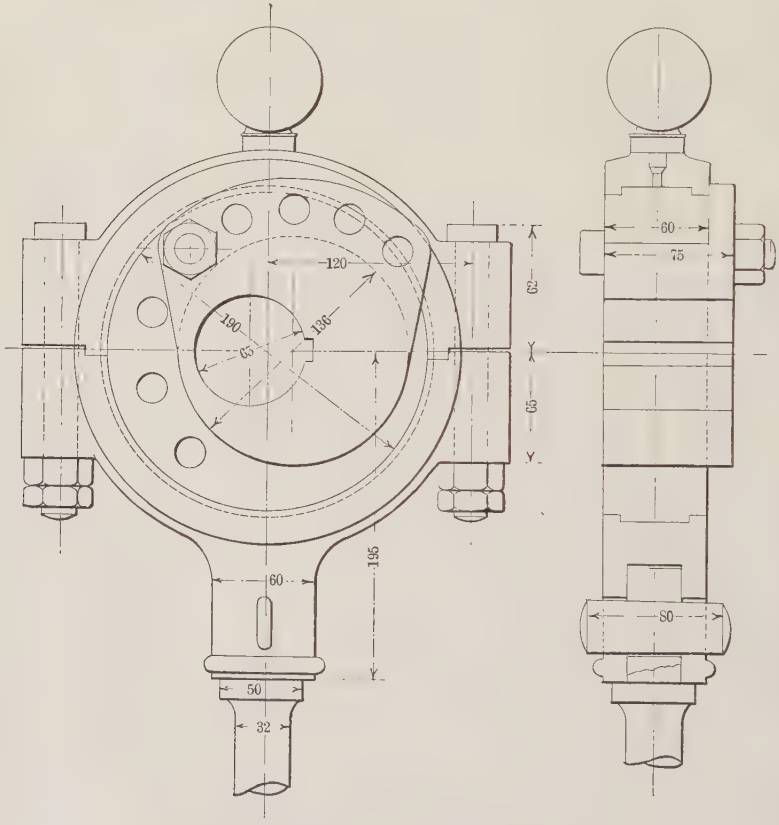


FIG. 4.—ECCENTRIC OF HYDRAULIC JIG.

In jigs treating grains larger than 10 mm., the ore falls on sorting-tables of perforated iron sheets. The jigs have two discharge-pipes, one for each compartment, and the division between the compartments is raised only as high as the pipe, to allow free movement to the upper layer. The first pipe discharges principally a mixture of galena, barite, and cerussite; the second discharges smithsonite and calamine. Sorting on the outside tables gives finished products.

The jigs with five compartments, for sands between 2 and 10 mm., discharge at the same time by pipe-discharge and hutch. A bed of iron disks—the waste from punching-machines—spread on the screen gives the resistance necessary for the separation of the sands and secures the continuous production, above the bed, of a layer of ore, which is forced out through the pipe-discharge as it is formed. To close the spigot, a stopper of some sort is employed, or else a bend, which can be turned upwards when it is desirable to stop the outflow. The screens have perforations larger in diameter than the maximum diameter of the sands, and the products from the pipe and from the hutch of the same compartment have nearly the same composition.

Table II. shows the principal features of the jigs in use at Monteponi.

TABLE II.—*Details of Construction of Jigs at Monteponi.*

	Jigs for Coarse Grains (2 compartments).			Jigs for Sands (5 compartments).			
Class treated, mm	20–30	14–20	10–14	7–10	5–7	3.5–5	2–3.5
Free width of compartments, mm	450	450	450	450	450	450	450
Free length, mm	750	750	750	500	500	500	500
Diameter of the holes in the screen, mm	10	8	6	10	8	6	4
Diameter of the iron disks which form the bed, mm.	.....	.....	.....	12–16	10–14	8–10	5–8
Stroke of the piston, mm	40–50	35–45	30–40	20–35	20–30	16–24	15–20
Number of strokes per min.	100	110	120	125	130	150	180
Approximate clear water per min., liters	140	100	75	50	45	40	40
Power consumed, h.p.	1.25	1.1	1	1.5	1.5	1.5	1.5
Material treated per hour, kg.	500	450	400	300	300	300	300
Diameter of the discharge-pipes, mm	51	38	32	25	20	16	13

All these jigs are directly fed by the vibrating-screens, and perform continuous work. The mixed products from the jigs for sand—for instance, the mixtures of galena, barite, cerussite, and smithsonite—are separated by closed jigs, with one compartment of 0.45 by 1.20 m. free surface of screen, giving beds of different ores, which can be removed by hand at intervals.

3. *Oscillating-Tables.*[6]

For sands below 2 mm. to 0.05 mm. the oscillating-table has been in use since 1898. This apparatus is well known also in

[6] French Patent No. 22,874; Belgian Patent No. 142,072; English Patent No. 22,374; German Patent No. 105,097; Austrian Patent No. 1,354.

other countries, since the Fried. Krupp Grusonwerk bought the patent and introduced it into almost all mining regions.

The oscillating-table is built in two types : one for fine sands below 2 to 0.5 mm., the other for sands of 0.5 down to 0.05 mm. They are identical in principle. The former is shown in Fig. 5. A rectangular table is placed horizontally in the direction of the movement, and slightly inclined in the other direction. It rests on six inclined springs, and receives an oscillating motion from an eccentric, exactly like the vibrating-screens; the table is covered with linoleum. Its inclination may be regulated during the progress of the work by wedges placed between the table and the frame, which rests on the springs. The mixture of water and sand from the hydraulic

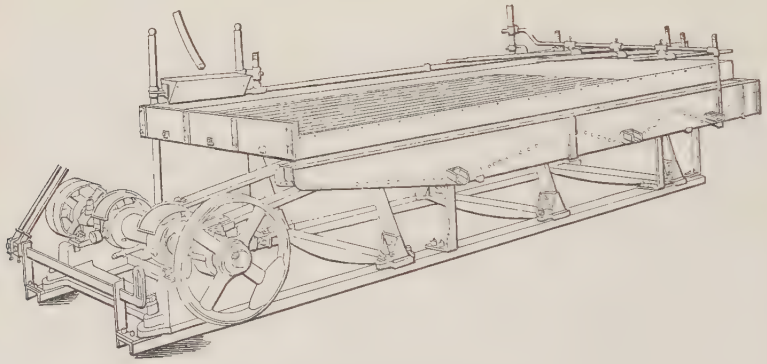


Fig. 5.—Oscillating-Table for Sands from 2 to 0.5 mm.

classifier is distributed by a short longitudinal hopper to the upper angle at the side of the eccentric, while the water flows away transversely. The grains are discharged on the table, running in parabolic lines, according as their specific gravity is greater and their diameter smaller. The spray-pipe placed at the upper side of the table pours out a slender stream of water which holds the grains suspended. Lengthwise grooves depressed in the linoleum prevent a too rapid fall of the heavy grains (without stopping the fall of the waste), and force them under the short spray-pipes placed at the end opposite to the hopper, where they are divided into groups of different character and specific gravity, and pushed towards the outlet.

The second type, or small oscillating table for sands finer than 0.5 mm., is trapezoidal in form, and has no spray-pipe at

the outlet; and the hopper at the entrance is replaced by a screen placed a few centimeters above the table, with which it oscillates. The purpose of this screen is to remove the excessively large grains, and to deliver the material evenly. This delivery is made first upon a raised section (*A*, Fig. 6), less inclined than the rest of the table, *B*, so as to hold the grains, while the accompanying water flows away transversely. The two sections, *A* and *B*, carry semicircular grooves, which diminish in depth towards the side of the outlet. The grooved area is limited by a parabolic line, as shown in Fig. 6.

This table serves also to treat the mixed products from the larger table of the first type, and all the other intermediary fine products. For this work, a screen is used with perfora-

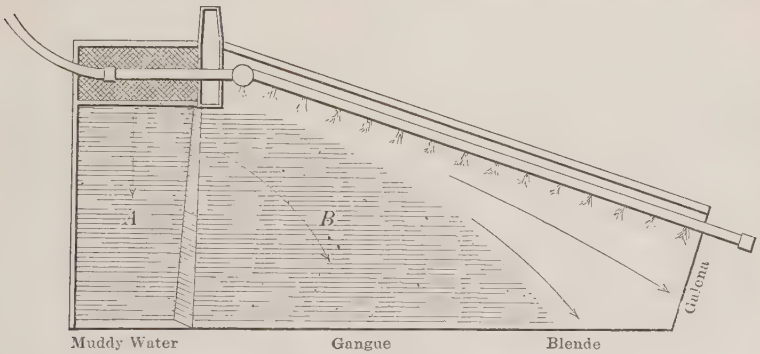


FIG. 6.—OSCILLATING-TABLE FOR SANDS FINER THAN 0.5 MM.

tions of 1 mm., corresponding to the maximum diameter of the grains which the table can treat successfully. The details of the two types of oscillating-tables are given in Table III.

TABLE III.—*Data of the Two Types of Oscillating-Tables.*

	Large Type.	Small Type.
Length of table, m., ...	3.5	2.25
Width of table, m., ...	1.5	1.40–0.5
Oscillations per min., ...	340	350
Amplitude of oscillations, mm., . . .	16–18	12–15
Water used per min., liters, ...	50	10–15
Necessary force, h.p., ...	0.75	0.5
Dry weight of material treated per hr., kg., .	400–600	200–400

4. *Comparison Between Hydraulic Jigs and Tables.*

In all old mills, sands of 1 to 2 mm., and even below 1 mm., are treated by hydraulic jigs with suction. The defects of this system are numerous. In the first place, sizing on screens,

and still more by trommels, of grains smaller than 2 mm., is difficult and far from exact; and the work of the machines for classification is costly and delicate. Suction-jigs for fine sands never give well-finished products; for below 2 mm. the pipe-outlet which serves to regulate the thickness of the bed, while maintaining on the screen of the jig a constant layer of ores of the same composition as that which sifts through the screen, cannot be used. The metallic value, or average specific gravity, of the ore which sifts gradually diminishes from one end of the jig to the other, without any sharp separation between the ores of different quality. The oscillating-table has the additional advantage of using less power in order to obtain better products, as can be seen by comparing the results of the two systems:

	Oscillating-Table.	Hydraulic Jig.
Net weight of material treated per hour, kg.,	400-600	300
Clear water per min., liters,	50	40
Motive-power per machine, h.p.,	0.75	1.50

When we consider that half of the products of the suction-jig are submitted to an extra concentration or separation, we see that the advantages of the oscillating-table are increased to about 50 per cent., and, apart from the best results in work, there is also considerable economy in installation.

5. *Belt for Treating the Slimes.*[7]

When argillaceous ores are treated, there are found in the last products from the hydraulic classifiers very fine ores, which run over the tables without sinking into the grooves. These fines have a diameter below 0.02 mm., and would go to enrich the slimes in the settling-ponds if there were no way to separate them. The method employed for this purpose serves, also, to recover the useful ores which might be drawn away by the water accompanying the products of the tables and jigs. It consists of a rubber belt, slightly inclined transversely, 0.60 m. wide, stretched over two drums, of which one serves to give the motion and the other the necessary tension. Every 60 cm. it is supported by rollers with regulated inclination, so as to have the belt almost horizontal at the side of the

[7] *Oesterreichische Zeitschrift für Berg- und Hüttenwesen*, vol. xlii., No. 35, p. 421 (Sept. 1, 1894).

entrance of the slimes, while the inclination is progressively increased at the side of the outflow of the products. Fig. 7 shows such a belt.

The pulp, reaching the belt through a rubber pipe, with almost no velocity, flows out on the belt, on which it deposits the solid particles, leaving the clear water to flow away. The motion of the belt carries the deposits to the water-sprays, which force them to the edge of the belt, making the lighter portion flow out with the water. The results are different products, some finished and some middlings, which can be treated on a second belt.

The force necessary to operate the slime-belt is merely that required to turn the belt on the pulleys without a load. A belt

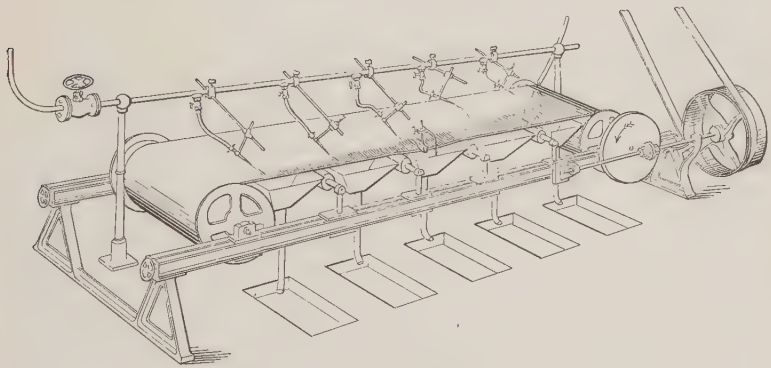


FIG. 7.—BELT FOR TREATING SLIMES.

4 m. in length treats 40 liters of slime, and requires 60 liters of clear water per minute.

According to the degree of concentration of the slime, more or less material can be treated on the belt, up to a maximum of 240 kg. of dry material per hour. The average is 100 kg., for, generally, concentration of the slime is avoided, so as to prevent losses by being carried away.

There are belts made two or three times the length, with two or three sections of introduction for the slime to be treated.

The belt does the same work as the Linkenbach revolving buddle; but it has the advantage over the latter of occupying less space. The life of a rubber belt, carefully managed, is about two years.

6. *Separation of the Middlings.*

Middlings from the process of concentration may be divided into two classes:

1. Mixtures of ores of too close a specific gravity to be easily separated at the first operation; for instance, cerussite and barite, blende and pyrite, calamine and limonite; or certain mixtures of sufficiently different specific gravity, but produced in the work of concentration, which are further treated either to remove a mineral which is found in the raw material in too small a quantity to be directly concentrated, or to take away from the waste all trace of useful mineral. Such are mixtures of galena with cerussite and barite, zinkiferous limonite and dolomite, as well as the ferruginous calamine and dolomite at the calamine-mill of Monteponi.

2. Mixed minerals which require a previous breaking to separate them.

As observed in connection with the hydraulic jigs for coarse grains, the mixtures of the first class are separated by stratification on the closed hydraulic jigs with one compartment, removing the products by hand, and layer by layer, as soon as stratified.

The fine-grained mixtures which contain waste are usually concentrated in a special section of the washery, provided with suitable classifying- and separating-apparatus.

Separation of the mixtures of the second class begins with crushing, more or less extreme, according to the nature of the material. The machines for crushing used in Sardinia are the stone-breaker, the rolls and the ball-mill. Of the first two types in Sardinia there is nothing special to be said.

The ball-mills used are chiefly the Krupp and the Ferraris.

Ferraris Mill.—The Ferraris wet ball-mill[8] possesses the advantages of great simplicity of mounting and small requirements of space and power for the same capacity. The steel plates which form the lining do not need to be adjusted, being held in place by the lateral steel walls and the sand formed by the crushing of the ores. There being no central shaft, large lumps of ore can be introduced into the mill, and workmen can easily enter for repairing and cleaning.

[8] United States Patent No. 726,521, Apr. 28, 1903.

The mill is made in two forms: one for coarse grinding (from 5 to 15 mm.), the other for fine grinding (from 0.5 to 5 mm.). The following description of the first form may serve for both, except as to the differences mentioned below.

The mill consists of a drum supported on four carrier-wheels and driven by a spur-gear securely fixed to the drum, which engages with a spur-pinion keyed to the counter-shaft. The drum is divided by an annular perforated partition into two compartments. The larger or crushing-compartment is $61\frac{3}{8}$ in. in diameter by 30 in. long. It is lined with manganese-steel plates with projecting ribs, and contains about 1,000 lb. of forged steel balls 4 in. and 6 in. in diameter. The smaller or screening-compartment, about 10 in. in length, is divided into a series of pockets by means of a cone projecting into the crushing-compartment, and a series of radial partitions extending therefrom. The periphery of this compartment is open, and is surrounded by a screen of the desired mesh. The material passing through the screen falls into a housing surrounding the lower half of the screening-compartment.

The ore to be crushed is fed into the crushing-compartment with the water, and, when reduced to pieces smaller than the holes in the annular partition, passes through into the screening-compartment, where the material which is fine enough passes out through the screen, and the oversize is elevated by the radial partitions until it slides back on the surface of the cone into the crushing-compartment, where it undergoes further crushing.

The drum is rotated at 20 rev. per min., and requires from 5 to 6 h.p. at its full load. The capacity at this speed on quartzose ore, broken by crusher to pass through a 2-in. ring, is approximately:

Mesh of screen, ...	12	16	20	30
Capacity, in tons per 24 hr., ...	35	30	25	20

The weight of the mill, including balls, is approximately 7.5 tons.

In this type, the peripheral plates are detached from the inner walls of the drum, leaving between them and the projecting bars a space of 12 mm., through which the water carries into the sizing-compartment the grains below 12 mm. In the second or fine-grinding form (Fig. 8), the peripheral steel plates are close

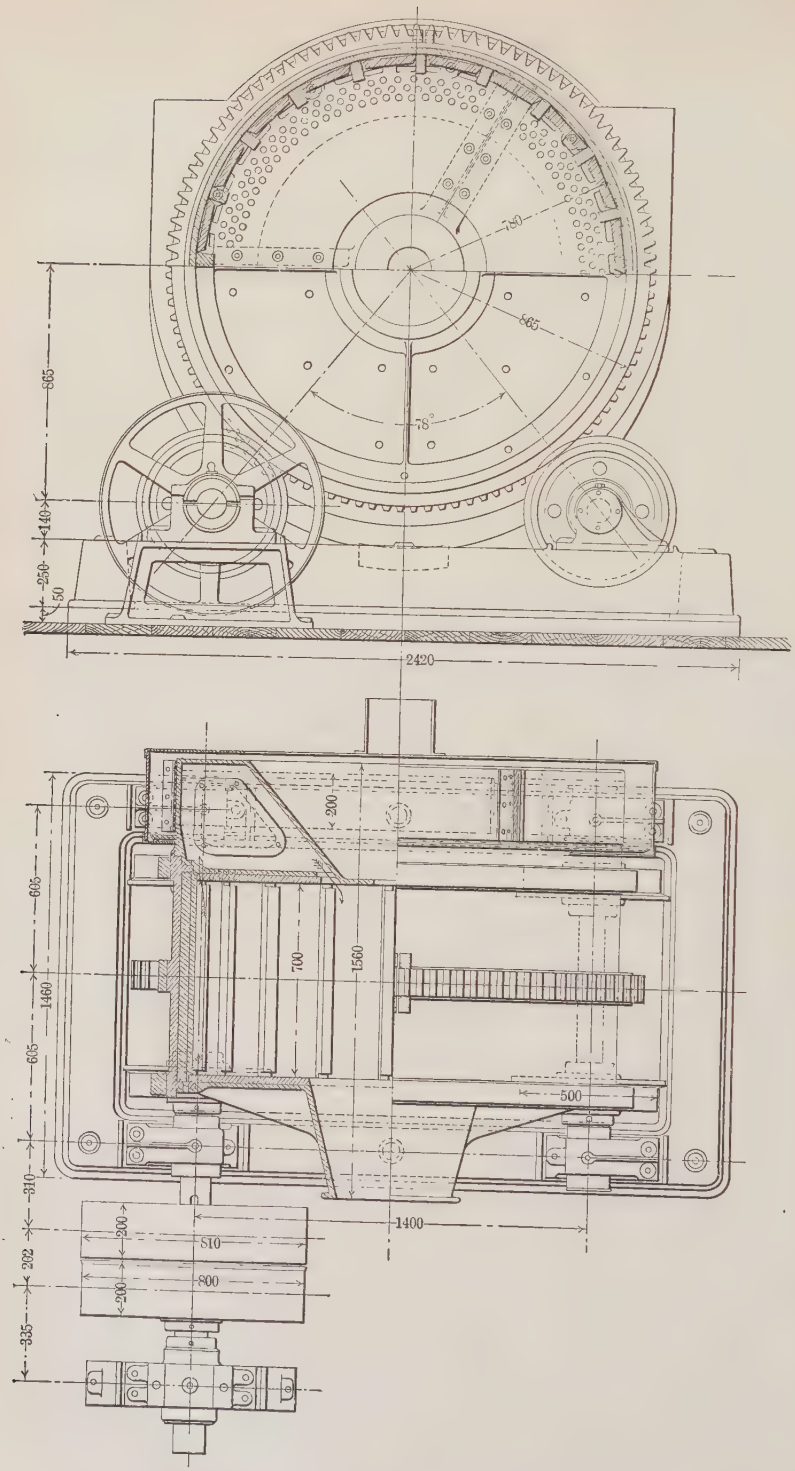


FIG. 8.—FERRARIS BALL-MILL FOR FINE GRINDING.

to the inner walls of the drum, and the water with grains below 10 mm. runs out through holes in the walls which divide the ball-chamber from the sizing-chamber. In both forms the screen is placed at the periphery of the sizing-chamber, and the material rejected by the screen is raised by the radial partitions to the point where it can slip over the exterior surface of the cone and return to the crushing-chamber.

A Ferraris ball-mill requires 7 h.p., with 20 rev. per min., and 80 liters of water per min. The quantity crushed per hr. depends on the quality of the ore and the size. In general, the product is greater from brittle ores like quartz than from tough minerals like diabase. A quartzite mineral in large pieces is crushed to 3 mm. at the rate of 4 tons in 3 hr., or 1.33 tons per hr. If the ore has been broken beforehand to 50 mm., 1.5 tons per hr. can be crushed to an average size of 1.5 millimeters.

The broken ore is sent to the separating-machines after having been sized, if a screen of more than 2 mm. in size is used. In this case the sizing is accomplished by the vibrating-screen. If the crushing is pressed below 2 mm., hydraulic classifiers are applied to the pipe which carries the water and sand, as described above.

In the mill at Monteponi for the fine crushing of mixed ores, the first hydraulic classifier feeds a jig of five compartments; the others feed the Ferraris oscillating-tables.

At the Rosas mine, there are five ball-mills forming five sections. The ball-mills receive the material which has been broken by the stone-breaker to 2 in. and crush it to 2 mm., at the rate of 1.5 tons per hr. per mill. But diabase impregnated with blende and galena is found to be very difficult to crush.

Each section is composed of one ball-mill, two jigs and three oscillating-tables. There is one special section, composed of a distributing-trunk, a classifying-pipe, and eight oscillating-tables, to treat the middlings from the five crushing-sections.

7. *Magnetic Separation.*

The mixed products of the ores of zinc and iron are treated by a reducing calcination, followed by magnetic separation.

Calcination.—This is performed in the well-known Oxland cylindrical furnace, of 1.30 m. exterior diameter, 1 m. inte-

rior diameter, 10 m. length, and 6.2 per cent. slope. The furnaces make an average of 15 rev. per hr., and serve to calcine the calamine below 15 mm. in size, as well as the mixtures of calamine and iron, to be later separated by the electro-magnet. In preparing the mixtures for magnetic separation, 2 per cent. of carbon is added to the ores for reducing action.

At Monteponi there are three rotating-furnaces, which gave, in 1906, the following results:

Hours of work in the year,	15,800
Weight of crude material introduced into the furnaces,	15,137.6 tons.
Weight of calcined product,	12,184.8 tons.
Total consumption of lignite,	2,296.85 tons.

It should be observed that the fuel is a lignite rich in ash, which gives 23 per cent. of clinkers; it is burnt on a barred grate in a thick layer with injection of air and steam under the grate. The fuel is partly gasified, and the gas burns in the furnace with the air heated around the hearth and on the hot calcined charge which falls from the furnace. A rotating-furnace can calcine a ton of crude ore, and give 773 kg. of calcined product per hour. The total fuel-consumption is 145 kg. per ton of crude ore, or 188 kg. per ton of calcined product.

In 1906, the cost of calcination per ton of calcined product, was:

	Francs.
Fuel,	3.2500
Hand-work,	0.7376
Steam and motive-power,	0.5000
Oiling and repairs,	0.2651
Total,	4.7527
Per ton of crude material,	3.825

Preparations are in progress to install in Sardinia new revolving-furnaces, which will have a tubular boiler between the furnace and the chimney, and thus avoid the expense for motive-power and the injection of steam. In this case the calcination of a ton of crude ore will cost only 3.5 francs at the most.

Magnetic Separators.—At Monteponi there are two installations of magnetic separators, one with six electro-magnets, rubber belts which carry the classified ore, and a large cross belt which removes the iron-ore. In order to distribute the material to the six electro-magnets, it is raised by a bucket-

elevator, and sized by a vibrating-screen into six classes; that is to say, 0–0.5; 0.5–1; 1–2; 2–4; 4–6; and 6–10 mm. The material over 10 mm. is crushed and returned after crushing to the magnetic separator.

The distance between the belt which carries the ore and the poles of the electro-magnet varies from 20 to 40 mm. An apparatus with six electro-magnets treats on an average 1 ton per hr., and requires 2 h.p. and a current of 6 amperes at 110 volts.

After separation from the iron, the zinkiferous product is dressed on closed jigs to remove the calcined dolomite and the small amount of lead-ore which it contains.

In the year 1906 one of these magnetic plants treated 6,373.97 tons of calcined material containing 25.98 per cent. of zinc. After separating the iron, and jigging, a marketable product was obtained of 2,264.12 tons with an average of 40.87 per cent. of zinc, representing 66.47 per cent. of the zinc in the original calcined ore and the removal of 17.31 per cent. of iron.

The remainder goes into the middlings products, which are set aside, and into the tailings from the jigs.

The iron oxide contains 10 per cent. of zinc, which cannot be removed without resorting to chemicals. To enrich still further the valuable calcined calamines, single and portable magnetic separators are used at Monteponi.

One of these drum separators[9] takes 2 amperes at 110 volts, and treats between 500 and 600 kg. of material per hour.

Another, with scissors arrangement, is similar to the multiple separators, but is stronger and can use up to 20 amperes. There are adjustable branches and an oscillating transporter.

8. *Calamine Mill, Monteponi, Sardinia.*

The ores treated consist of the economic minerals, calamine, smithsonite, and limonite, with some galena, cerussite, siderite, and sphalerite. The gangue is limestone and dolomite, with some barite. The smithsonite and galena are very compact, and, upon crushing, remain largely in the coarse products, while the calamine and cerussite are very friable and break up

[9] *Engineering and Mining Journal*, vol. lxxxii., No. 24, p. 1129 (Dec. 15, 1906).

into fines. The galena carries about 0.2 per cent. of silver, but the cerussite contains very little silver.

Ore from the mine-cars is dumped to (1).

(1) Grizzly having 80-mm. openings between the bars. From the mine; delivers oversize, via hopper, to (2) and undersize, via hopper, to (3).

(2) Picking-table. From (1); delivers calamine to market, mixed zinc-iron-lead ore to (12), limonite to market and waste rock to dump.

(3) Two Ferraris waving-screens, each having three screening-sections, with holes 14, 20 and 30 mm. in diameter respectively. From (1); deliver material on 30 mm. to (7), material from 20 to 30 mm. to (6), material from 14 to 20 mm. to (6) and material from 0 to 14 mm. to (4).

(4) Two Ferraris waving-screens, each having three screening-sections, with holes 5, 8 and 10 mm. in diameter respectively. From (3); deliver material on 10 mm. to (6), material between 8 and 10 mm. to (8), material from 5 to 8 mm. (8) and material from 0 to 5 mm. to (5).

(5) Two Ferraris waving-screens with two screening-sections, with holes 1.5 and 3 mm. in diameter respectively. From (4); deliver material on 1.5 mm. to (8) and through 1.5 mm. to (9).

(6) Twelve 2-compartment jigs. From (3) and (4); deliver mixed lead- and zinc-ore to (12), calamine to market and tailings to dump.

(7) Wire picking-belt. From (3); delivers rich calamine to market, ferruginous calamine to market, poor zinc-iron middlings to (12), limonite to market and waste rock to dump.

(8) Sixteen 5-compartment jigs. From (4) and (5); deliver lead-zinc middlings to (12), rich calamine to market, ferruginous calamine to (32), limonite to market, poor iron-zinc middlings to storage and tailings to waste.

(9) Hydraulic classifier. From (5); delivers spigots to (10) and overflow to (11).

(10) Four 5-compartment jigs. From (9) and (10); deliver cerussite to market, lead middlings to (10), calamine to market, rich iron-zinc middlings to (26), poor iron-zinc middlings to storage and tailings to waste.

(11) Six Ferraris waving-tables. From (9); deliver cerus-

site to market, calamine to market, iron-zinc middlings to (26) and tailings to waste.

Recrushing Department.

(12) Ferraris wet ball-mill. From (2), (6), (7) and (8); delivers to (13), crushes through 8 millimeters.

(13) Ferraris waving-screen, having three screening-sections, with holes 1.5, 3 and 5 mm. in diameter respectively. From (12); delivers material on 1.5 mm. to (14) and material through 1.5 mm. to (16).

(14) Three 5-compartment jigs. From (13); deliver middlings to (15), calamine to market, ferruginous calamine to (32), rich iron-zinc middlings to (19), poor iron-zinc middlings to storage and tailings to waste.

(15) Four-compartment jig. From (14); delivers lead-ore to market, lead-barite middlings to (35), calamine to market, iron-zinc middlings to (26) and tailings to waste.

(16) Hydraulic classifier. From (13); delivers spigot to (17) and overflow to (18).

(17) Five-compartment jig. From (16) and (17); delivers cerussite to market, lead-zinc middlings to (17), calamine to market, ferruginous calamine to (32), iron-zinc middlings to (26) and tailings to waste.

(18) Two Ferraris waving-tables. From (16); deliver cerussite to market, calamine to market, iron-zinc middlings to (26) and tailings to waste.

Auxiliary Middlings Department.

(19) Ferraris waving-screen, having two screening-sections, with holes 5 and 8 mm. in diameter respectively. From (14); deliver material on 5 mm. to (20) and material through 5 mm. to (22).

(20) Four 5-compartment jigs. From (19) and (22); deliver lead middlings to (21), rich calamine to market; ferruginous calamine to (32), iron-zinc middlings to (26), poor middlings to storage and tailings to waste.

(21) Four intermediate jigs run intermittently and discharged by hand-skimming. From (20); deliver cerussite to market, lead-barite middlings to (35), calamine to market, poor iron-zinc middlings to (26) and tailings to waste.

(22) Ferraris waving-screen, having two screen-sections, with holes 1.5 and 3 mm. in diameter respectively. From (19); delivers material on 1.5 mm. to (20) and material through 1.5 mm. to (23).

(23) Hydraulic classifier. From (22); delivers spigots to (24) and overflow to (25).

(24) Two 5-compartment jigs. From (23) and (24); deliver cerussite to market, zinc-lead middlings to (24), calamine to market, rich iron-zinc middlings to (26), poor iron-zinc middlings to storage and tailings to waste.

(25) Three Ferraris waving-tables. From (23); deliver cerussite to market, calamine to market, iron-zinc middlings to (26) and tailings to waste.

Magnetic-Separation Department.

(26) Revolving cylindrical furnace. From (10), (11), (15), (17), (18), (20), (21), (24) and (25); delivers to (27).

(27) Ferraris waving-screen, having six screening-sections, with holes 0.5, 1, 1.5, 2.5, 4.5 and 6 mm. in diameter respectively. From (26); delivers material to (28).

(28) Ferraris magnetic separator. From (27); delivers limonite to market, and non-magnetic tailings coarser than 2 mm. to (29) and finer than 2 mm. to (31).

(29) Three intermediate jigs run intermittently and discharged by hand-skimming. From (28); deliver limonite to market, calamine to market, middlings to (30) and tailings to waste.

(30) Intermediate jig run intermittently and discharged by hand-skimming. From (29); delivers calamine to market, middlings to storage and tailings to waste.

(31) Three 4-compartment jigs. From (28) and (31); deliver limonite to market, middlings to (31), rich calamine to market, ferruginous calamine to (32) and tailings to waste.

(32) Revolving cylindrical furnace. From (8), (14), (17), (20) and (31); delivers to (33).

(33) Ferraris waving-screen, having seven screening-sections, with holes 0.5, 1, 1.5, 2.5, 4.5, 6 and 10 mm. in diameter respectively. From (32); delivers to (34).

(34) Ferraris magnetic separator. From (33); delivers calamine to market and limonite to market.

(35) Revolving furnace for decrepitating barite. From (15) and (21); delivers to (36).

(36) Ferraris waving-screen, having six screening-sections, with holes 0.5, 1, 1.5, 2.5, 4.5 and 6 mm. in diameter respectively. From (35); delivers material on 6 mm. to market, from 4.5 to 6 mm. to (38), from 1 to 4.5 mm. to market, from 0.5 to 1 mm. to (37) and below 0.5 mm. to market.

(37) Three 4-compartment jigs. From (36) and (37); deliver lead-ore to market, mixed lead-ore to (37), barite to market, calamine to market and tailings to waste.

(38) Intermediate jig run intermittently and discharged by hand-skimming. From (36) and (38); delivers lead-ore to market, lead-zinc middlings to (38), calamine to market and tailings to waste.

Primary Gold in a Colorado Granite.

BY JOHN B. HASTINGS, DENVER, COLO.

(New York Meeting, February, 1908.)

TEN miles from Hartsel, near Antelope springs, in Park county, Colorado, there is a large area of unconsolidated lake-beds, which are interesting because at least a part of the lacustrine sands contains in the aggregate an immense amount of gold.

During 1906, one of the arms of this dessicated lake was fairly well prospected with shafts and cuts by an Eastern company to test its value, as favorable amounts of gold had in some instances been found upon it. The claims of the company contain about 5,000 acres of the beds; but this is only a small portion of them, a south branch from a main body, which is at least 7 miles in diameter, with other and very wide lateral gulches, paralleling the one herein described on the east and west. A certain amount of digging and sampling was done in the outside areas; but the result was evidently unsatisfactory, since the work was discontinued.

This section of Colorado has not been studied geologically, and the age of the ancient lake is unknown. It is probably Quaternary. I spent two weeks on the ground, in September

and October, 1906, but was quite absorbed in the sampling. I looked, however, for terraces on the rim-rock and other evidences of movement and erosion, but did not find anything to denote an earlier age than the Quaternary. The granite rim-rock may be Archean; S. F. Emmons says that some of the granites in central Colorado are as old as that.

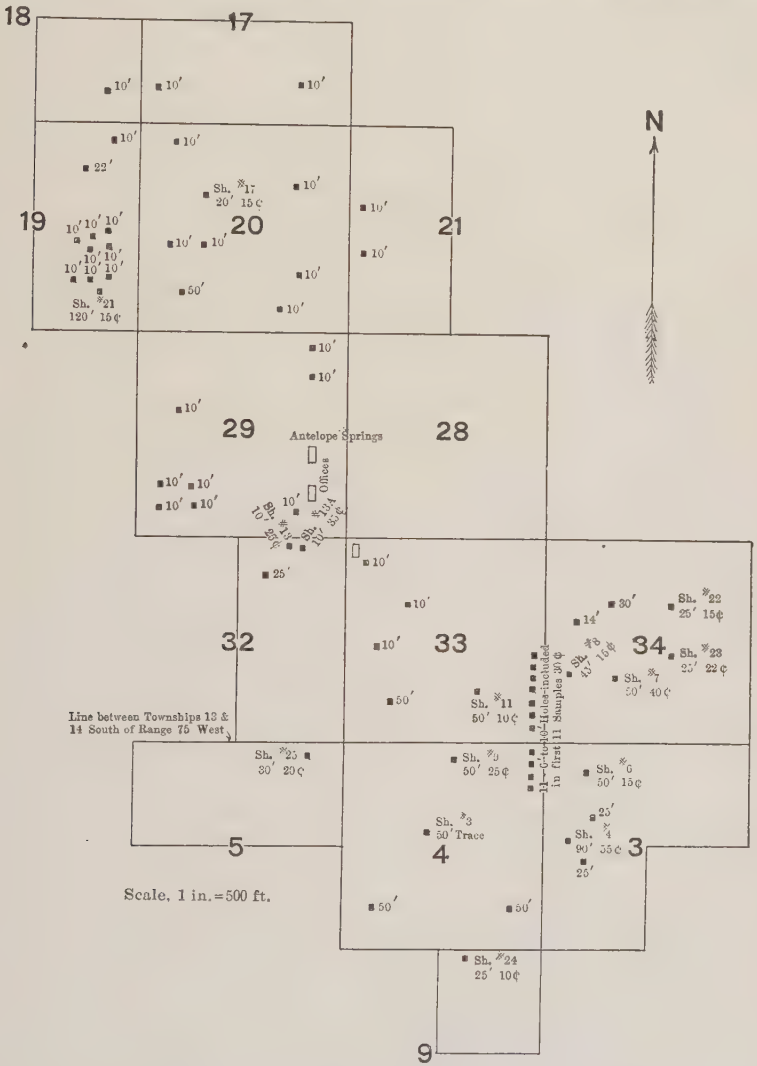
The rim-rock as observed is a gray quartzose biotite-granite, except a small area in Sections 3 and 34, where it is a felsitic (?) breccia. The breccia is silicified by solfataric action at the corner near Shaft No. 7, in Section 34; but this action is older than the sands and does not extend to them. The shaft averaged 50c. per ton, more than twice as much as the rest. This result is excluded from any of the estimates of the total value of the ground. West of the camp, immediately beyond a narrow rim of granite, there is a reddish rhyolite outcrop.

I carefully noted the character of the sands, and found them an ordinary mixture of quartz, feldspar, and biotite; brown when dry, bluish when wet; and concluded that on the whole they were derived from degradation of the granite rim-rock. The feldspars are usually entirely altered, but the mica—black, shining, unoxidized biotite—is remarkably persistent. This suggests that the detritus is volcanic ash; but the biotite in the granite rim-rock has the same characteristic resistance to decomposition. Even in the granite subjected in the past to solfatarism, which produced a complete alteration of the feldspar, and a partial one of the quartz, the bright biotites remained. Except in some unusual places I did not see any suggestion in the arrangement or character of the deposit that it was tuff. In these places the material was very fine, and may well be granite slime. The beds range from clay to coarse sand, but not gravel.

A distinction had been made by prospectors between the brown and the blue sands; but they are the same in composition, and there is no unconformity between them, or anything to indicate that they were laid down at different periods, or under different conditions. The brown is the dry sand slightly stained with iron oxide; the blue, the damp sand near permanent water-level.

Interest had also been felt in the "fines," or that portion of sand which as dug would pass a 4-mesh screen. These fines are

the looser material in the homogeneous mass of sand, or the part loosened by abrasion in breaking the ground, and are not seams of sand in gravel or breccia. From my samples, the



NOTE.—Samples were taken from only the numbered shafts.

FIG. 1.—SKETCH-MAP OF TEST-SHAFTS NEAR ANTELOPE SPRINGS, PARK COUNTY, COLO.

fines were screened and quartered down separately, and were found to carry just the same amount of gold as the sands.

I watched carefully, but could not find any areas or strata of

concentration. There is but a small range of value in the samples (from 10c. to 40c. per ton); and the total average is 20c. per ton. The only high assays were from Shaft No. 7, at the silicified area already noted, and they were excepted from the estimate.

The map, Fig. 1, shows that the shafts and cuts are scattered so as to prospect the ground thoroughly. Besides 11 pits, from 6 to 10 ft. deep, in Sections 4 and 33, there are 55 shafts, aggregating 1,375 ft. in depth, of which 32 are 10 ft. deep, and two are respectively 90 and 120 ft. deep. Altogether the average depth is 25 ft. Some of these shafts were full of water, or otherwise inaccessible. From shafts and dumps I was able to sample satisfactorily 15 of them, aggregating 650 ft. in depth. The shallow pits in Sections 4 and 33 were also sampled.

It would have been an easy matter for some one to salt such sands as were sampled; hence, much precaution was observed. In sampling the shafts, six continuous cuts were made, four on the sides and two on the ends. Each sample consisted of 6 or 7 ft. in height of these cuts, so that three samples for assay would be made from a shaft from 20 to 25 ft. deep. Before making the sample-cuts, a much larger mass of the material was stripped down to obtain fresh surfaces. The samples taken weighed 3.5 tons; the material first removed to obtain these weighed probably 14 tons. To find out whether the fines (or sands which as broken would pass a 4-mesh screen) contained more gold than the coarse, each of the samples (4,037 lb. of the original weight of 7,062 lb.) was screened through a 4-mesh screen, resulting in 2,723 lb. of oversize and 1,314 lb. of undersize; 50 samples had been taken, given 50 oversize and 50 undersize assays; and both classes averaged the same (20c. per ton), exclusive of Shaft No. 7. It was also hoped that this division would help to expose any preliminary salting. If the samples had contained gold in paying quantity, further investigation would have been made, but being worthless, the matter was dropped.

The 100 samples, 50 oversize and 50 undersize, were given to three Denver assayers for as careful determination as ordinary prices would allow. One office returned a "trace" on 33 of the samples, and the assayer, on inquiry, said that the trace

was about 25c. or less; the other two assayers, Messrs. Stephen Rickard and J. W. Richards, averaged 20c. on the other 66 samples, exclusive of Shaft No. 7. These gentlemen used respectively duplicate half assay-tons and whole assay-tons in the determinations. A synopsis of these results is given in Table I. and Fig. 1. The value is pretty evenly distributed throughout the ground. From these returns it seems safe, in figuring on the "ore in sight" (which in this case is not ore at all), to estimate a block of ground, 5,000 acres in area, and 50 ft. deep, at 25 cu. ft. to the ton, equals 435,600,000 tons, at 20c. per ton, equals $87,120,000, and this is only a small portion of the lake-bed area; but let me quickly add that even if my sampling was correct, and this gold is existent, it is as elusive as the gold in sea-water. It would cost, including losses, several times as much as its value to obtain it.

I took a "grab" part of each of the 50 pulverized undersize samples, making a composite sample, and panned it. The slimes assayed 40c., middlings 10c., and concentrates a trace, per ton. Too much confidence must not be placed on such results from a single sample, no matter how carefully taken. It is a series of samples, balancing errors, that counts. I also took three samples, each one-eighth of an assay-ton, as if for assay, from the same slimes, middlings, and concentrates, and examined each microscopically, with from 40 to 70 diameters, expecting to find colors of gold, but failed to do so. The gold of Snake river takes 1,000 to 1,200 colors to make a cent, and would have been easily visible under the circumstances. Further attempts to "chase down" this gold are left to a wiser man.

None of the shafts are down to bed-rock, but there is nothing to indicate that the bed-rock will be richer than the sands above it, although the sands are probably richer than the rim-rock from which they are derived. No concentration of any kind was found in the beds; and, judging from panning-tests, none of the gold is free. I did not make these tests, but while the development of the ground progressed, the superintendent and the foreman panned constantly without finding any appreciable colors. They undoubtedly tried carefully and skillfully, being anxious to find gold. I thought the lack of free gold, and consequently of concentration, was due to the peculiar resistance to

alteration of the iron silicates, and that if these silicates had released the gold the resultant exhibit would have been similar to the gold of Snake river, Idaho; but the higher value of the slimes, containing a modicum of unaltered biotite, has confounded this idea.

TABLE I.— *Value of Gold in Sands.*

Description.	Oversize. Ct.	Undersize. Ct.
Composite sample from four of the six pits in Sec. 33; the individual samples were unfortunately put together,	40	40
Composite sample from pits Nos. 5, 6, and 7 in same place,	20	30
Composite sample from pits Nos. 8, 9, 10, 11 in same place,	20	20
Shaft No. 4, full of water, sample of half the dump,	60	50
Shaft No. 3, inaccessible. Sample of one-quarter of the dump,	trace.	
Shaft No. 13,	20	30
Shaft No. 13A,	50	20
Shaft No. 7, average of 6 samples,	30	50
Shaft No. 22, two samples of bottom 14 ft. to timbers,	20	15
Shaft No. 23, two samples of bottom 14 ft. to timbers,	20	15
Shaft No. 8, four samples of bottom 28 ft. to timbers,	20	15
Shaft No. 6, three samples of bottom 21 ft. to timbers,	15	20
Shaft No. 9, six samples of bottom 38 ft. to timbers,	25	25
Shaft No. 24, two samples of bottom 14 ft. to timbers,	10	10
Shaft No. 21, full of water, sample of one-quarter of the dump, average of three from bottom of trench, 27 ft. long, 2.5 ft. deep, 1,200 lb.,	15	10
Shaft No. 11, 50 ft. deep, tightly timbered for 27 ft., three samples from bottom 23 ft.,	10	10
Shaft No. 17, 20 ft. deep, two samples covering 8.5 ft., skipping timbering, from bottom to surface-soil,	15	15
Shaft No. 25, 30 ft. deep, two samples of bottom 10 ft.,	20	20

I chipped off 250 small pieces from four areas of the rim-rock, 0.75 mile apart, on the west side of the ground of the company. At each place these pieces were taken from granite outcrops, dotted over 5,000 sq. ft.; the sample was carefully assayed and yielded only a trace of gold. The resulting button from two assay-tons looked very nicely under a Coddington lens, but it did not appreciably move the scales, though it was said they would weigh down to 10c. per ton if one assay-ton were used.

In taking the chips, slips showing quartz-seams and aplitic intrusions were avoided. To find out about such material, I took 50 chips from an island, rising out of the lake-bed, 1.5

miles south of the camp and springs. These scattered croppings, 200 by 100 ft., seemed to have been originally an aplitic type of the general granitic country; they are now somewhat secondarily silicified. The sample assayed 10c.; three others tried at the same time gave no gold. The assayer said that others besides myself had brought in Hartsel lake material, and the sands always contained gold, though sometimes the country-rock did not.

The story is interesting as indicating one of those virginal sources of supply from which nature has elsewhere gathered the metals for payable deposits. Gold in granite is not unique. W. P. Blake[1] cites S. F. Emmons, G. P. Merrill, J. E. Spurr, and himself, as discoverers. It was found by A. Simundi in the general mass of the granite of the Carson district, at Silver City, Owyhee county, Idaho,[2] and by others in various places.

It is from such sparsely mineralized areas that the metallic values are thought by some geologists to be gathered into pay-veins, where both gold and quartz result from magmatic differentiation, as opposed to solfatarism. The dividing-line between deposition by the two phenomena is probably a matter of personal opinion or bias. So far, observation of such processes in action covers only both extremes—dikes with a small percentage of water, and attenuated waters, the solid contents of which would do no more than form a film on their channels of passage.

Reasoning upon an ordinary commercial quartz-vein as a product of magmatic segregation, there is only required a concentration of silica from the original magma of three into one; but if the vein carries $7.50 per ton in gold, and the original magma 10c. per ton, and 5c. is left behind, there would be a gold-concentration of 150 into one.

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[1] Gold in Granite and Plutonic Rocks, *Trans.*, xxvi., 290 (1896).

[2] *Report of the Tenth Census*, vol. xiii., p. 54 (1880).

Origin of Pegmatite.

BY JOHN B. HASTINGS, DENVER, COLO.

(New York Meeting, February, 1908.)

THE occurrence of such a large amount of gold in the Hartsel granite, even though the surmised existence of similar areas is not new, brings freshly to mind the pegmatite type of magmatic differentiation lately brought again to attention by occurrences at Silver Peak, Nev., described by Spurr: and in Alaska, described by both Spurr and Brooks. In both localities granitic rocks or dikes are thought to pass into metalliferous quartz-veins.

An interesting association in this connection is the occurrence of diamonds in residual pegmatitic clay of San Juao de Chapado, Brazil, described by Derby:[1]

(P. 142) "Bodies of pegmatite are quite common in the older rocks of Brazil, both in the diamond regions and elsewhere, occurring not only in the gneiss and granite, but in the schistose series as well. Those that have been examined are dike-like in their mode of occurrence and granitic in composition."

(P. 143) "The question of the eruptive or secretionary origin of pegmatites has long been a subject of discussion among geologists. . . . The recent studies . . . seem to have clearly established that most if not all of them are essentially eruptive masses, though possibly modified in some way by aqueous agencies. Even before becoming acquainted with the literature of the subject this view had seemed to me to be the only acceptable one as regards the typical pegmatites of Brazil."

What is pegmatite? J. F. Kemp says:[2]

". . . veins or dikes—it is an open question which is the more correct term—are met formed of very coarsely crystalline aggregates of the same minerals that constitute granite. These are called pegmatite and in them is the home of graphic granite, the curious intergrowth of quartz and feldspar, such that a cross fracture of the blades of quartz suggests cuneiform characters. . . . In regard to the larger veins or dikes it seems improbable that true igneous fusion could have afforded such coarsely crystalline aggregates, and so we are forced to assume such abundance of steam and other vapors, *i.e*, mineralizers, as to almost, if not quite, imply solution."

[1] Brazilian Evidence on the Genesis of the Diamond, *Journal of Geology*, vol. vi., No. 2, pp. 121 to 146 (Feb.–Mar., 1898).

[2] *Handbook of Rocks*, 1st ed., p. 31 (1896).

A. Geikie,[3] quoting Fouqué and Michel-Lévy, and de Lapparent, says:

"Where the quartz and feldspar of a granitic rock have crystallized in one common direction, one within the other, the structure is pegmatitic where visible to the naked eye, and micro-pegmatitic where the aid of a microscope is needed."

J. P. Iddings's definition in part is:[4]

"The amount of individualized inclusions in a crystal is often so great, that one may speak of a mutual penetration of two or more materially and morphologically different substances. Such a mutual penetration of quartz and acid feldspars is especially common; it presents a peculiar appearance, characteristic of certain members of the quartz-porphyry group, and is the so-called micropegmatite or granophyre structure. The same intergrowth is frequently observed between members of the feldspar group (microcline, albite, orthoclase) in the older massive rocks, where it is usually controlled by rigid mutual crystallographic relations. The basic massive rocks also exhibit similar phenomena, as, for example, when the larger porphyritic augites are so filled with apatite, magnetite, mica, nepheline, haüyne, etc., that the augite substance only forms a cement, as it were, for the different minerals. This structure has been called poicolitic."

Its microscopic features are, Hague and Iddings say:[5]

"*Pegmatoid.*—The structure produced by the intergrowth of two or more minerals, usually quartz and feldspar, in such a manner that when seen in thin section they appear as groups of grains of more or less regular form, each group having one optical orientation throughout. It is similar to the structure in graphic granite."

The same properties, with other and familiar names, are described by E. Haworth[6] in discussing the structure of the holocrystalline ground-mass of porphyries, who says:

"The following is taken from the lecture notes of Prof. G. H. Williams: . . . In the second place a granular effect may be produced by the complete interpenetration of two individual crystals of the same size. In this case—due to the simultaneous crystallization of the two minerals from the magma—all the parts of the same individual, no matter what the size or shape, must have exactly the same optical orientation, and must hence extinguish the light between crossed nicols together. Such a structure is termed, according to the particular form it assumes, micropegmatitic or granophyric."

[3] *Text-Book of Geology*, 2d ed., p. 109 (1885).

[4] Rosenbusch, *Microscopical Physiography of the Rock-Making Minerals*, p. 19 (1888).

[5] On the Development of Crystallization in the Igneous Rocks of Washoe, Nevada, *Bulletin No.* 17, *U. S. Geological Survey*, footnote, p. 14 (1885).

[6] A Contribution to the Archæan Geology of Missouri, *American Geologist*, vol. i., No. 6, p. 367 (June, 1888).

This structure is quite common; quotations like the following are often seen. N. H. Darton,[7] after examining thin sections of granite from Montana areas, says:

"Micrographic intergrowth of quartz and feldspar is common, and secondary minerals frequently occur along the cleavage cracks of the latter."

Arrhenius, as quoted by Vogt,[8] discussing pegmatites, begins at the beginning, and, presuming an interior gaseous earth magma, says, in effect, that so far as we know these gases are primarily homogeneous, and if they contain water (vapor), on ascension as plutonic magmas and consequent degree of cooling would separate into less and more aqueous segregations, especially at contact with the cool inclosing rock, the separation becoming ever more marked, and the aqueous part more mobile. Then:

"By reason of the greatly superior mobility of the aqueous solutions, as compared with the magma, these segregations may send out branches in the form of the finest apophyses. The solution in aqueous gas now gradually cools, and one substance after another separates from it. By reason of the great mobility of the solution, and its consequent strong capability of diffusion, the minerals (provided the cooling be not too rapid) are segregated in large crystals, such as characterize a so-called pegmatitic structure. Gradually, also, the constituents which longest retain a gaseous form—such as water and carbonic acid—escape."

This seems closely akin to solfatarism, and the product, if seasoned with metallics and garnished with walls, should make a desirable vein. In descriptions of pegmatites it is sometimes hard to understand just what is really meant regarding "solutions," now that the term is applied to ordinary liquid magmas.

Pegmatites are plentiful in Norway and Sweden, and Prof. Hjalmar Sjögren[9] describes them:

"These pegmatites are to be considered as secretions with a low temperature of crystallization, deposited from aqueo-igneous solutions in contraction-fissures due to the cooling of the surrounding rocks."

Waldemar Lindgren's[10] version, about pegmatites in general:

[7] Geology of the Bighorn Mountains, *Professional Paper No.* 51, *U. S. Geological Survey*, p. 19 (1906).

[8] Problems in the Geology of Ore-Deposits, *Trans.*, xxxi., 133 (1901).

[9] The Geological Relations of the Scandinavian Iron-Ores, *Trans.*, xxxviii., 766 (1908).

[10] Relation of Ore-Deposits to Physical Conditions, *Geological Congress, Mexican Meeting*, 1906.

" . . . made by aqueo-igneous solutions, pegmatite dikes . . . pegmatite formation evidently requires very high temperature, and the conditions are decidedly deep-seated."

This shows a slight difference of opinion; in fact, to get a fair knowledge of the habits of pegmatites and theories of their origin it is necessary to read a number of descriptions; the following, arranged about chronologically, are the most important that I have seen in recent American literature, and probably cover quite well the diversities of occurrence, and are given either verbatim, in quotation marks, or without them, but closely following the language of the authors:

Sterry Hunt[11] makes a strong case for so-called pegmatitic veins in eastern Canada and the northeastern United States.

He, in common with other geologists at that time, thought them segregation-veins, or concretionary, as he preferred to call them, and describes dikes and veins of granite, and says the latter are distinguished by containing boron, fluorine, phosphorus, etc., and by successive deposition generally in open fissures, showing persistency of the banded structure, each consisting of different minerals or association of minerals deposited slowly and successively from aqueous solution.

For instance:

(P. 194) "At other times pure vitreous quartz forms one or both walls, or the centre of the vein, or else is arranged in bands parallel with the sides of the vein, and sometimes a foot or more in thickness, alternating with similar bands consisting wholly or in great part of orthoclase, or of an admixture of this mineral with quartz, having the peculiar structure of what is called graphic granite, or else presenting a finely granitoid mixture of the two minerals, with little or no mica, and with small crystals of deep red garnet."

The widths varied from a few inches to 60 ft. These veins also contained the rarer minerals. Experiments of Schafhautl and Wohler, which show that quartz and apophyllite may be dissolved in heated water, under pressure, and recrystallized on cooling, are also alluded to.

Coming after Hunt, and perhaps the first of American geologists to apply microscopic study to pegmatites, was G. W. Hawes.[12]

[11] Granites and Granitic Vein-Stones, *Chemical and Geological Essays* (1874).

[12] *Geology of New Hampshire*, vol. iii. (1878).

Describing the granites of New Hampshire:

(P. 201) "Another of the hornblende rocks . . . is very peculiar in its microscopic structure. Macroscopically, this rock is fine in texture and dark in color, and its individual constituents cannot be determined. Under the microscope its thin sections show that the dark constituent is greenish hornblende, though it is partially altered into epidote and chlorite; but the circumstance of note is, that many of the minute grains of feldspar are inlaid with quartz in the manner peculiar to graphic granite. The rock is hence a kind of microscopic pegmatite. The figures that are formed are often of rare perfection. Michel-Lévy found this character in quite a number of French granites. . . . The pegmatitic character of a rock he regards as evidence of the simultaneous crystallization of quartz and feldspar."

Referring to the immense beryls found in New Hampshire:

(P. 68) "The beryls are found in granitic veins. The veins are easily recognized by the very large crystals of quartz, feldspar, and mica, which are the constituents of ordinary granites; and the general presence of beryls in them is interesting as substantiating the theory of their formation. These granitic minerals occupy large fissures, and it is thought that water, which had filtered through the surrounding rocks, and which, under a high pressure, and at a high temperature, had become saturated with their soluble constituents, deposited these great crystals of the various minerals in these fissures, until they were finally filled with this extremely coarse granitic mixture. In this way the rarer elements, such as glucinum, which exist in such minute amounts in the surrounding rocks, became concentrated in these veins, forming the beryls that are so common there."

This is a lucid description of the former lateral-secretion theory as applied to pegmatites, which was in general belief 20 or more years ago: since then so much has been learned of the character of veins that perhaps these views are not held at all now in the United States.

G. H. Williams,[13] describing the associated rocks of the crystalline belt near Baltimore, Md., says:

(P. 14) "Huge veins of a coarse-grained pegmatite, which both in their form and in their structure closely resemble intrusive rocks, are also frequently found intersecting the schists and gneisses. As far as my observation extends, the character of these granites seems quite independent of the rock in which they occur. Whether in the lightest, most acidic gneiss or in the darkest hornblende-schist, the coarse-grained aggregate of muscovite, microcline, albite, and quartz appears in all respects the same."

(P. 25) "Accidental minerals are rare in the Baltimore gabbros. At some localities vein-quartz—evidently a secondary infiltration into existing cracks and fissures—is abundant. Long crystals of black tourmaline, forming radiating groups sometimes two feet in diameter, are found imbedded in this quartz. . . .

[13] The Gabbros, etc., in the Neighborhood of Baltimore, Md., *Bulletin* No. 28, *U. S. Geological Survey* (1886).

"Associated with this quartz, west of Mount Hope Station and along Gwynn's Falls, north of the Liberty road, are many blocks (float? J. B. H.) of a coarse-grained muscovite granite or pegmatite, which is to all appearances identical with that occurring so abundantly in the gneisses. This rock is composed of a flesh-colored microcline, white albite, quartz, and muscovite; garnet and tourmaline are also not rare constituents. The occurrence of this pegmatite, so rich in alkalies, in the center of the gabbro area, may be regarded as a fact in favor of its exotic nature so far as it shows that this rock is not dependent upon the nature of the inclosing mass, as might naturally be expected to be the case with segregation veins."

The following description by Prof. W. P. Blake[14] has an added interest because the pegmatite is an ore (?) deposit, and in structure forms a connecting link with tin-veins.

Describing the Black Hills tin-veins: The Etta is a granitic mass in fine-grained slates, about 200 ft. to 100–150 ft. The mass crystallization is extremely coarse; slabs of pure feldspar from 12 to 20 in. long and masses of white quartz several feet thick are abundant:

(P. 692) ". . . the outer portions, next to the country rock, being characterized by a band or belt of dark-colored mica, alternating in places with muscovite in large plates. This is succeeded by massive quartz, with irregular bunches of massive albite and of orthoclase feldspar, together with enormous crystals of spodumene and irregular bunches of a dense aggregation of small crystals of mica and albite, forming a kind of greisen-rock, an albitic greisen, in which cassiterite is abundantly disseminated in small grains and partly formed crystals."

(P. 695) "In the numerous tin-veins and tin-ore-bearing granitic dikes of the Black Hills tin-region, the phenomena of occurrence and association indicate that all of the minerals of the dikes—the quartz, feldspar, spodumene, mica, beryl, columbite, tantalite, phosphates, and other associates of the cassiterite, were contemporaneous in origin. The tin-stone is apparently as much a part of the mass as the mica or quartz. It was, to all appearance, present when the whole mass assumed its crystallization. All the constituents of the dike appear to have crystallized from a semi-fluid or pasty magma, in which the elements were free to arrange themselves from one side of the dike to the other and to crystallize out slowly."

No crustification, alteration, replacement, or solution, no fluorine. The rude parallel structure seems due to different conditions of pressure or temperature next to the walls while the fissure was full of the unsolidified constituents.

H. W. Fairbanks,[15] on southern California pegmatites:

"On the eastern slope of this hill is an enormous pegmatite vein, carrying a very interesting set of minerals. This vein is twenty or more feet wide, and dips

[14] Tin-Ore Veins in the Black Hills of Dakota, *Trans.*, xiii., 691 to 696 (1884–85).

[15] *11th Annual Report, California State Mining Bureau*, p. 94 (1890–92).

west at a small angle. There are masses of great size of almost pure mica and feldspar, or quartz and feldspar—in the latter case very fine specimens of graphic granite have been formed. Near the southern end of this vein is a deposit of lepidolite mica, 10 feet thick at the widest part, and appearing in detached bodies for several hundred feet. It is fine-grained and shows a pale purple color. In places it is pure, in others filled with large radial aggregates of pink tourmaline (rubellite). Some of the aggregates are a foot across, others are long and slender, with arborescent forms. North of the main deposit it is found in quartz in fan-shaped aggregates, the crystals being more than a foot long, but greatly decomposed. Black tourmaline is abundant in the pegmatite surrounding the lepidolite, but in poor, brittle crystals. Garnets are also to be found in places. The vein as a whole is inclosed in the diabase."

G. H. Williams on the origin of Maryland pegmatites:[16]

(Footnote, p. 675) "The term 'pegmatite' was first suggested by Haüy in 1822 for those regular intergrowths of quartz and feldspar which are now designated 'graphic granite.' In 1849 Delesse extended its use over all the very coarse granites. He was followed by Naumann in his *Lehrbuch der Geognosie*, and since that time the word pegmatite has been generally adopted for this entire group of coarse-grained granitic rocks, and even for the corresponding equivalents of other plutonic masses, as for instance syenite-pegmatite, diorite-pegmatite, gabbro-pegmatite, etc."

(P. 675) "Another important group . . . largely developed in Maryland . . . embraces the very coarse-grained aggregates of quartz and feldspar, with more or less mica, variously known as pegmatites . . . they are among the youngest of the products of granitic intrusion, and hence show little or no effect of dynamic metamorphism. . . .

"Pegmatites are . . . abundant all the way from Newfoundland to Alabama. . . . Many of the districts in eastern United States which have become mineralogically famous are great pegmatite dikes."

(P. 676) "Élie de Beaumont . . . accepting in the main the igneous and intrusive origin of pegmatites, introduced an important addition in assuming water and other mineralizing agents as necessary factors in their formation. He correlated the pegmatites with the other phenomena so common in the peripheral regions of granitic districts, or, as he called it, 'granite aura' (the penumbra of Von Humboldt). De Beaumont, while assuming granitic emanations as necessary for the crystallization of the coarse-grained granites, is careful to distinguish between them and the banded concretionary veins formed by substances dissolved in circulating heated waters. . . .

"The intrusive theory of the origin of pegmatite, with the aid of water and other mineralizers as important factors, has been recently advocated by J. Lehmann and by Brögger, and may be regarded as the most acceptable for all those masses which are in intimate association with larger plutonic intrusions."

(P. 678) "Only by assuming that they have attempted to cover by a single explanation a number of similar rocks which are genetically distinct is it possible to account for so many competent observers having arrived at such diverse conclusions regarding the origin of pegmatites. . . . The writer's studies in Maryland have led him to believe that both segregation and intrusive pegmatites, quite similar in appearance, appear side by side."

[16] General Relations of the Granitic Rocks in the Middle Atlantic Piedmont Plateau, 15*th Annual Report, U. S. Geological Survey*, pp. 657 to 684 (1893–94).

Illustrative of the two forms of pegmatite in the Piedmont Plateau, there is, first, an immense amount of white vein-quartz, seaming and penetrating the schists in all directions, from minute isolated eyes to large lenticular masses, and veins continuing for a considerable distance, occasionally containing original tourmaline and flesh-colored feldspar, and, rarer, mica, etc., all formed by segregation from the inclosing rock-sheared gneisses. Secondly, there are the dikes composed of quartz, microline, albite, muscovite, and occasional small patches of red garnet (no biotite was observed), similar to but more acid than the main mass of the granite, that are plainly eruptive. Their size and abundance are proportional to their nearness to some eruptive granite mass with which they essentially agree in chemical and mineralogical composition, and they are quite independent in character of other rocks in which they may be inclosed. They are not, as a rule, either drusy or symmetrically banded, though frequently finer grained toward the edge.

(P. 684) "The writer therefore interprets those pegmatites which by their mode of occurrence and association strongly indicate an igneous character as the products of the residual and therefore most acid portion of a granite magma highly charged with water and other mineralizing agents. Such a siliceous material, in a state intermediate between fusion and solution, has been injected into fissures and there crystallized into very coarse-grained aggregates, not necessarily through any great slowness of this process, but rather in virtue of the aid to crystallization afforded by the abundance of mineralizers present."

The very fine illustrations accompanying the paper show typically appearing dikes.

Messrs. Crosby and Fuller[17] (from whose work the following are extracts and principles), complementing the work of Williams, have together with that author supplied the views in the leading treatises on the subject in the United States, for the past ten years, and perhaps still supply them, though largely supplemented by the monograph on Silver Peak, Nevada, by J. S. Spurr.

(P. 327) "The modern conception is that in a broad view the pegmatites are igneous rocks, but it is the part which water has played in their formation that has so strongly differentiated them from other igneous types . . . the chief purpose of this paper is a contribution to the aqueo-igneous theory."

Pegmatite has been employed to designate both the macro-

[17] Origin of Pegmatites, *Technology Quarterly*, vol. ix., No. 4, pp. 326 to 356 (Dec., 1896.)

and micro-pegmatitic structure, also plutonic rocks distinguished by a gigantic scale of crystallization. Pegmatites range from highly acidic (quartz) to ultra-basic varieties, and differ from normal plutonics texturally, rather than mineralogically; we herewith deal with the acid pegmatites.

(P. 328) "The essential species include quartz; the acid feldspars—orthoclase, oligoclase, albite, and microcline—the last two being especially characteristic; and the more acid micas, including muscovite and lepidolite, and, less characteristically, biotite."

The accessory minerals are numerous and include many rare species. The quartz, as in granite, contains numerous inclusions of water and carbon dioxide, in one instance in proportion of 1–1, indicating crystallization under enormous pressure. We abundantly confirm Williams that typical pegmatite masses pass into pure quartz and ordinary quartz-veins. We disagree with his belief that their origin is different, and feel that quartz-veins are the possible end-product of differentiation that yields pegmatites. At Narragansett bay mica-schists contain typical pegmatite veins from 5 to 60 ft. wide; a mile east, veins in the schist are quartz, with some feldspar but no mica, while on another island they are quartz alone. Pegmatites occur in all formations, but—

"In every pegmatite district there is one normal plutonic rock of essentially similar but slightly less acid composition, with which the pegmatite is most intimately associated, into which it may often be traced, and from which it has evidently been derived."

The pegmatites most often in the district studied penetrate schists composed chiefly of silica and alumina, and have absorbed portions of them, producing as a result more quartz and muscovite, the latter being an aluminous silicate. The pegmatite-mines of New Hampshire are in such schists. Perhaps tourmaline pegmatites generally were formed the same way. Pegmatites apparently carry more unusual minerals than the parent rock, but this is probably due to differentiation into tangible local accretions, as the total contents of magma and dike agree chemically. While the texture of the pegmatites may be as fine as that of the parent plutonic, they are on the whole characterized by their coarseness, beryls from a foot to a yard in diameter, spodumenes from 10 to 30 ft. long, feldspars from 10 to 20 ft.,

and other crystallized silicates occur, beautifying the best cabinets. The smaller veins, from 2 to 20 ft. wide, are often coarser-grained than the greater ones. The minerals crystallize in the same order and manner in the pegmatites as in the plutonic—tourmaline and other basic species, biotite, muscovite, basic feldspar, acid feldspar, and quartz. The earlier minerals are approximately idiomorphic, the quartz allotriomorphic. In the vugs, however, the order is changed, except for tourmaline and ultra-basic minerals, and albite is often seen in tabular crystals implanted with muscovite.

(P. 334) "Notwithstanding the lateness of its crystallization, the quartz not infrequently forms immense vitreous masses, and these often pass into veins of quartz, or quartz and accessory feldspar, intersecting the normal pegmatite as well as the country rock, thus testifying clearly to the extremely acid character of the magma residuum."

Continuous crystallization is abundant in all the larger masses, that of the later species having begun before the earlier ceased. Intergrowth of quartz and feldspar is frequent, and perthitic intergrowth of the feldspars similar in structure to graphic granite also occurs, but the ideal pegmatitic development of the latter is rare.

(P. 334) "Rosenbusch has sagaciously correlated the pockets and druses with the miarolitic structure of the normal plutonics—a feature of like significance, but developed, like the crystallization, on a grand scale. . . .

"The pegmatite masses, like true dikes, are frequently fine-grained next the wall, becoming rapidly coarser toward the center."

The fine-grained portion sometimes grades into the parent plutonic wall-rock without line of demarcation. A cross-section from a well-defined pegmatite vein, 6 in. wide,—

(P. 335) "is made up of two sharply defined and symmetrical bands or layers of albite (clevelandite), with the tabular crystals set edgewise to the walls and a median band of smoky quartz; while springing directly from either wall and penetrating both the albite and quartz are numerous slender prisms of green and red tourmaline.

"The disposition of the tourmalines noted above is highly characteristic; and may be observed in many massive or unbanded veins; showing that this comb-structure and a distinct banding are not necessarily correlative, although they are undoubtedly of like significance, testifying with equal distinctness to successive crystallization."

Vugs are sparing here but characteristic in New Hampshire,

where they are large enough to admit of one well-formed quartz-crystal a yard in diameter. These great pockets have indefinite positions and are never peripheral in the vein; in other districts they occur in flatly lenticular form, indistinguishable except mineralogically from pockets of ordinary veins. Wall-rock fragments are common in the pegmatite; the inclusions, schist especially, have a frayed out, skeleton-like appearance, suggesting solvent action; the wall-rock is similarly affected and impregnated with tourmaline and allied minerals. The pegmatites no doubt occupy spaces of discission, and occur in wall-like, lenticular, and horizontal sheets, but a very large proportion are too irregular for any one term.

(P. 337) "They very commonly conform closely with the structure planes of the schistose rocks (interbedded veins); but they also intersect the bedding planes or lamination at all angles; and may in general be assumed to follow lines of least resistance to the disrupting force."

The occurrences in the granite have "parallel sides," generally more "definite angles and general linear character" than in the schist, where the intrusion has evidently often formed for itself spaces of dissolution. The authors say, in effect:

(P. 340) The rare minerals, coarse crystallization, banding and comb-structure, with tourmalines, etc., normal to the walls, inclusions of water in quartz, etc., the pockets and druses of the pegmatites favor an aqueous origin; but the order of crystallization (except in pockets), the completely idiomorphic crystals, the graphic structure and the finer crystallization next the walls, are at least more suggestive of igneous contacts; broken tourmalines, etc., inclined tourmalines, inclusions of carbon dioxide, immense size of many of the veins, evidence of solution of the walls, orientation of inclusions, demand . . . important modifications of the aqueous process.

The granite country is uniform and devoid of pockets.

Brögger is cited that gigantic crystallization and irregular composition are due to slowness of consolidation, the walls being first heated; also that pegmatite dikes are often far away from the parent eruptive.

The authors picture as a parent magma, a deep-seated granite boss cooling from the boundary inwards, and crystallizing with ever-increasing slowness, thus driving the siliceous waters to the center of the mass and there forming the giant crystals. The magma contracts with solidification, so the hardened outside crust of the boss and the beds on top may

fissure, and the central magma residuum enter these fissures, forming pegmatite. There should be a gradual transition from the normal granite to pegmatite, and it is thought that a syenite in the vicinity with feldspar crystals from 4 to 6 in. long may represent this stage. It is also thought that normal magma often penetrated hot schists and absorbed water from them, adding to its own liquidity, which enabled the formation of pegmatite. The pegmatites are believed to have been formed at great depth.

A. Mervyn Smith,[18] describing Indian pegmatites, says:

The veins range from a mere thread to 20 ft. wide, and occur parallel to the bedding of a mica schist. The veins are made up of amorphous masses of quartz, large crystals of pink orthoclase, and crystals or books of muscovite. When the walls are quartzitic the vein-stuff is nearly pure quartz; when highly feldspathic, pink feldspar crystals are the chief constituent; when micaceous, mica predominates. Such occurrences were probably responsible for the lateral-secretion theory.

F. D. Adams,[19] describing nodular granite from Pine Lake, Ont., says in effect:

The nodules were "schlieren" rich in silica and alumina, with boracic acid, and during the general cooling of the magma developed a spherulitic arrangement, often with tourmaline towards the center, and sometimes arranged themselves in a string, vein fashion. It is unknown how they first came into existence, but:

(P. 171) "We have, however, examples of such differentiation in granite magmas in the case of pegmatite veins, which at their extremities frequently run out into veins of quartz associated with a little tourmaline."

(P. 172) ". . . but the study of this occurrence shows that 'contemporaneous veins' of an acid character may be formed not only during the final stage of crystallization, as in the case of the hysterogenetic schlieren and the 'kluftblätter' of Reyer, but that highly silicious portions are sometimes segregated or differentiated out of a granite magma before crystallization, and that the banded structure often seen in pegmatites and other allied bodies and sometimes cited as proof of their aqueous deposition in preëxisting fissures is not necessarily so produced, but, as is now being generally recognized, may and usually does result from the primary crystallization of the cooling magma."

[18] Mica Mining in Bengal, India, *Mineral Industry*, vol. vii., p. 512 (1899).

[19] Nodular Granite from Pine Lake, Ont., *Bulletin of the Geological Society of America*, vol. ix., pp. 163 to 172 (1897).

J. F. Kemp,[20] cites a variation of the common minerals, near Port Henry, N. Y.

(P. 183) "Near the ore are also met coarse aggregates of hornblende, plagioclase, magnetite and quartz in the nature of pegmatites."

The ores are thought to be contact-deposits, though obscure; there is an almost total absence of sulphur.

H. B. Patton,[21] describes the strongly tourmalitic type in Colorado.

(P. 26) "The pegmatite veins of these foothills are usually composed of coarse granular aggregates of reddish microcline and quartz, with or without muscovite, and occasionally garnet. They frequently resemble segregation veins in that they shade off into the adjoining schists without any well defined vein wall. . . . It may be an intrusive dike, but in the opinion of the writer these and other pegmatite veins of the region are not of such origin."

The tourmalines occur:

(P. 21) "Beautiful lustrous black crystals, often two or more inches in diameter, have been obtained from these pegmatite veins."

(P. 22) "One may see . . . numerous veins . . . of quartz and feldspar containing the habitual black tourmaline."

First locality—an 18-in. vein of quartz and tourmaline:

"The tourmaline . . . is a fine grained schorl-like mass more or less banded with white vein quartz . . . tourmaline predominates over quartz . . . not as . . . grains or crystals, but rather in a dense felted mass . . . banded structure . . . is usually very marked . . .

"A thin-section . . . discloses . . . granular quartz, . . . prisms . . . and . . . grains of tourmaline, . . . a very little muscovite."

Second locality:

". . . a pegmatite vein about 10 feet wide. . . . The tourmaline, which is to be found only sparingly in the vein itself, occurs impregnating the schists at contact with the vein. . . . the schists . . . lose in places all traces of the original cleavage, and develop into aggregates of quartz and tourmaline to the entire exclusion of the mica. . . .

"The tourmaline has evidently been formed at the expense of the biotite."

J. F. Kemp,[22] on the Atlantic Coast granites in general, says:

[20] Geology of the Magnetites near Port Henry, N. Y., etc., *Trans.*, xxvii., 146 to 203 (1897).

[21] Tourmaline and Tourmaline Schists from Belcher Hill, Colo., *Bulletin of the Geological Society of America*, vol. x., pp. 21 to 26 (1898).

[22] Granites of Southern Rhode Island and Connecticut, etc., *Bulletin of the Geological Society of America*, vol. x., pp. 361 to 382 (1898).

(P. 372) "The pegmatites are present in great numbers, and at times attain very considerable size. . . .

"The commonest kind is a very coarse aggregate of red microcline, white natron-orthoclase, albite, and quartz, together with a little black, brittle biotite and occasional thin plates of ilmenite and masses of magnetite."

(P. 374) "The proportions of the several minerals in the pegmatites vary considerably, but in the normal specimens one might say that red microcline is most abundant and makes up about 50 per cent. of the whole. Natron-orthoclase and albite follow with about 25 per cent., then quartz, with perhaps 20 per cent., leaving 5 per cent. for all the rest. Quartz may, however, become much more abundant, and instances have been met, as in a large vein near Sachems head, which consists of quartz with but a few feldspar crystals distributed through it."

Then a pure and huge quartz-vein, with crustification, is described:

"As earlier stated, Prof. Dana observed that the quartz veins near Stony creek were later than the pegmatites, and it may well be that they mark the closing and fumarolic stages of the intrusive phenomena."

J. F. Kemp had also previously quoted Dana, that the granite (pegmatite) veins cut the quartz-veins when both occurred together.

J. E. Spurr,[23] describing the pegmatoid aplite on Fortymile and Birch creeks, Alaska, says, in effect:

There is about a complete absence of coarse pegmatite, but the fine-grained aplites change gradually in texture to coarse-grained aplites, or fine-grained pegmatites, without change in composition or structure. Sometimes, however, the quartz and feldspar separate, and there will be large bunches of pure quartz and, elsewhere in the dike, coarse-grained aplite, or fine-grained pegmatite. The predominating feldspar is sometimes plagioclase, sometimes orthoclase. In some of these same rocks first the feldspars contract into zones nearest the walls, and leave the central portion pure quartz; then the feldspars disappear altogether and the whole dike becomes solid quartz. The quartz, under the microscope, shows coarsely crystalline interlocking structure, the only minerals beside quartz being a little magnetite and pyrite.

A. H. Brooks[24] found similar occurrences in the Tanana and White River Basins, Alaska:

(P. 463) [Aplite is here used purely in a mineralogic and not in a structural sense.] "This is a massive rock, which when fresh is of an almost pure white color. It is usually medium grained, but is occasionally coarse enough to be called pegmatite. The typical aplite-granite of the region is composed of white feldspar with bluish vitreous quartz. The potash feldspar is orthoclase, sometimes micro-

[23] Geology of the Yukon Gold District, Alaska, 18*th Annual Report of the U. S. Geological Survey*, pt. iii., p. 230 (1896–97).

[24] Reconnoissance in the Tanana and White River Basins, Alaska, in 1898, 20*th Annual Report of the U. S. Geological Survey*, pt. vii., pp. 425 to 494 (1898–99).

cline, and with it occurs considerable albite. The thin sections of this rock which were examined showed a typical granite structure. It is entirely massive, but in a few instances a distinct parallelism of the mineral was noted, which was undoubtedly an original structure.

"Typically this is a quartz-feldspar rock, but often a little accessory muscovite is present, and it sometimes passes into a muscovite-granite. Apatite is a common accessory mineral, and in some specimens considerable tourmaline was noted. One phase of this rock is scantily sprinkled with small plates of biotite. This aplite-granite has a wide distribution in the White River and Tanana basins, and rocks of a somewhat similar character have been described by Spurr in the Fortymile and Birch Creek districts. I found them cutting not only the gneisses, but also rocks of younger age, as will be described hereafter. They are usually intruded as small dikes parallel to the foliation and bedding planes, but are occasionally found cutting the structural planes."

(P. 484) "*Quartz veins in the older series.*—In the descriptions of the rocks of the region reference has been made to the presence of quartz in the gneisses and the two oldest series of metamorphosed clastics. In the gneissic series the quartz veins are limited to the sheared and schistose phases whose foliation planes afford opportunities for the injection of the quartz-bearing solutions. On the Middle Tanana, where the two shear zones are equally developed, the quartz veins are found in both systems. In the quartz schists and associated rocks of the Nasina series of the Lower White the quartz veins reach a great development; they are found intruded parallel to the foliation, and very seldom cut across it. I noted in this series a close relationship between the quartz and the coarse pegmatite veins, and at a number of localities was able to trace transitions from one to the other. A similar transition has been noted by Spurr in the Birch Creek and Fortymile districts. The quartz veins are most abundant in the Tanana schists, whose finely fissile condition gave abundant opportunity for the penetration of the mineral-bearing solutions. The veins are more widely disseminated in the Tanana schists than in any of the other formations. The greenstone schists show very few quartz veins, for in many cases the rocks are too massive to afford any line of weakness along which the solutions could have penetrated."

J. Barrell,[25] on the pegmatites of the Elkhorn district, Montana:

(P. 518) Describes thin augite-syenite dikes, from a fraction of an inch to 18 in. wide, and as coarsely crystalline for such small dikes, the feldspars being sometimes an inch long. The constituents are orthoclase (largely microcline), augite, and mica. Abundance of water, low percentage of silica, and apparent high liquidity and slow cooling have allowed differentiation after intrusion, so that in patches, a foot or more in length, no augite is present, and surrounding this white nucleus are the panidiomorphic feldspars, thickly charged with augite crystals, making the magma blue-black.

[25] Microscopical Petrography of the Elkhorn Mining District, Jefferson County, Montana, 22*d Annual Report, U. S. Geological Survey*, pp. 511 to 550 (1900-01).

"The segregation has originated therefore, in place and before the beginning of crystallization."

The question of two ages of crystallization, one in the deep before extrusion, the other after the dike-magma is in place, is important, especially as to the larger crystallization; on it might depend a decision as to whether the latter was common to a large area of siliceous magmatic differentiation with a moderate supply of mineralizers, or to the influence of a later concentrated supply.

(P. 539) "The aplites, as the term is commonly understood, are rocks of a sugar-granular texture due to a fine-grained, simultaneous crystallization of quartz and orthoclase, the rock consisting of those components with little or no ferromagnesian minerals." . . .

"The aplites are commonly believed to be acid segregations out of the original magma" . . . [76 per cent. of silica.]

(P. 541) An interesting study was made of a thick intrusive sheet between quartz-monzonite and andesite. The aplite showed evidence of chilling against the andesite, and for an inch on the contact is coarse-grained, with quartz and feldspar poikilitically intergrown; then follows several feet of coarse-grained (2 mm. in diameter), characterized by a radiate poikilitic growth of quartz and orthoclase, with less biotite than on the contact.

"A thin section taken 52 inches from the contact showed certain crystals of orthoclase which became poikilitic only after half their growth was completed, indicating a change in the physical conditions at that period, owing to which the quartz and orthoclase could no longer separate themselves from each other during crystallization, and were of simultaneous growth."

At 20 ft. the aplite is normal.

T. L. Watson,[26] after examining the porphyritic granites of Georgia, concludes, after careful study, that the porphyritic feldspars were formed in place and are not a product of earlier crystallization in the depths; they were formed rapidly and are potash feldspars—in all of the sections some of the feldspars show micro-pegmatitic intergrowth with quartz.

A. H. Brooks:[27]

[26] On the Origin of the Phenocrysts in the Porphyritic Granites of Georgia, *Journal of Geology*, vol. ix., No. 2, pp. 97 to 122 (Feb.–Mar., 1901).

[27] Preliminary Report on the Ketchican Mining District, Alaska, *Professional Paper* No. 1, *U. S. Geological Survey*, p. 47 (1902).

"Pegmatitic and aplitic rocks. . . . These have more the appearance of segregations than true injections. In thin section a specimen from one of these was seen to be composed essentially of orthoclase and plagioclase more or less idiomorphically developed, and containing many gas or fluid inclusions, together with allotriomorphic quartz. Muscovite occurred as an accessory mineral. Another occurrence of pegmatite typically developed. . . . Here a coarse, white, pegmatitic rock cuts the greenstone schists. . . . In thin section these pegmatite dikes show an allotriomorphic intergrowth of plagioclase, orthoclase, and quartz. Many of the dikes have been mineralized and much altered. They are frequently brecciated and constitute the host of the ore body."

T. L. Watson,[28] on some Georgia pegmatites.

Stone mountain, residual of a larger mass, is about 700 ft. high (and, judging from the illustrations, three-quarters of a mile in diameter). Aplites are rather scarce and pegmatites common. The latter consist principally of coarse intercrystallizations of feldspar (orthoclase and microcline) and quartz, a little biotite and muscovite, occasionally red garnet and tourmaline; sometimes the pegmatites are replaced by pure quartz. Except for its white color, much finer grained texture, entire absence of biotite, and decreased muscovite, the aplite and granite are similar. There are small aggregates of tourmaline throughout the entire mass of granite, in centers of white areas of quartz and feldspar, from a fraction of an inch to several inches in diameter, sharply separated from the granite. These areas are interlocking quartzes and feldspar, totally similar to the granite feldspars; the tourmaline is thought to have been derived by fumarolic action from the feldspars.

(P. 186) "In the Stone Mountain pegmatites the dark minerals, mica, tourmaline, and garnet, are frequently concentrated along the central axis of the dike or vein, rather than distributed through the light-colored quartz-feldspar portions."

(P. 193) "While no distinct evidence bearing on the contemporaneous origin of the tourmaline aggregates in the granite with those of the pegmatite and the tourmaline veinlets, it seems reasonable to assume such contemporaneity.

"The very nature of the areas oppose the hypothesis of direct secretion out of the eruptive granite magma. On the other hand, the characteristic mode of occurrence and intimate relationship to certain other mineral species present, as shown both macroscopically and microscopically, make it reasonably certain that the tourmaline areas have resulted from fumaroles highly charged with boric acid acting on the feldspars and mica."

[28] On the Occurrence of Aplite, Pegmatite, and Tourmaline Bunches in the Stone Mountain Granite of Georgia, *Journal of Geology*, vol. x., No. 2, pp. 186 to 193 (Feb.–Mar., 1902).

A. C. Spencer,[29] on pegmatitic structure at Grand Encampment, Wyoming.

". . . aplite . . . is here used for white, fine-grained rocks which are composed principally of quartz and oligoclase feldspar. Pegmatite is used for related rocks which are more coarsely crystalline than the aplite and of pink or reddish color. The feldspar of the pegmatites is usually orthoclase or microcline."

"The geologic relations of the aplite have not been completely determined, but bodies of the rock composed of feldspar and quartz and having the form of dikes were found to pass by gradation into quartz-veins, a fact which suggests that they may not be true igneous intrusions in the same sense as the gabbro rocks, but rather that they have been formed through aqueo-igneous activity, an origin to which many pegmatites have been assigned.

"The coarse pegmatites. . . . Like the aplites, they are found to grade into quartz-veins."

Small amounts of gold have been found in the quartz-veins and secondary copper in crushed pegmatite.

J. E. Spurr,[30] on the Silver Peak pegmatites. While it is usually impossible to do justice to an author through extracts, it is very much so in this case, as they are from a compendious monograph bearing directly on the auriferous-quartz phase of pegmatitic differentiation.

On the subject of two periods of crystallization:

(P. 103) "The best crystallized muscovite occurs abundantly in cavities in granites and in pegmatites, associated with minerals like tourmaline, which is the result of the action of another mineralizer—boron."

(P. 105) Microcline almost always belongs to the second generation of crystallization, and zircon and pyrite always so.

C. W. Brögger is cited as placing pyrite and other sulphides in pegmatite dike-veins among the minerals formed at the second period of consolidation, when the action of water and other mineralizers was more potent in forming minerals than during the first period of consolidation, which was that of ordinary magmatic solidification; also, that while zircon is formed during the first period it also occurs among the minerals of the second period, and was probably formed by the action of mineralizers, presumably fluorine.

[29] Copper-Deposits of the Encampment District, Wyoming, *Professional Paper* No 25, *U. S. Geological Survey*, p. 41 (1904).

[30] Ore Deposits of the Silver Peak Quadrangle, Nevada, *Professional Paper* No. 55, *U. S. Geological Survey* (1905).

A. La Croix describes, in an alkali granite from Madagascar (frequently pegmatitic), allotriomorphic or idiomorphic zircon inclosed in the quartz, sometimes in the feldspar. He regards this zircon as later than all the minerals except quartz. It is distributed in the granite and not localized in veins, and regarded as the product of emanations contemporaneous with the final consolidation of the rock.

(P. 107) Partial alteration of the feldspar into sericite occurred when the magma was partially consolidated, before deposition of the remainder of the secondary minerals.

(P. 112) "The magma was a viscous fluid, less viscous than a normal granitic magma, but still capable of the rôle of intrusion. From this magma crystallized, besides some of the ordinary accessory granitic minerals, principally feldspar and quartz, the consolidation of the feldspar in general preceding that of the quartz. The proof that the magma penetrated the sedimentary rock mass in every pore, and that many of the larger intrusive lenses are of accumulated material which was intruded along small channels now often difficult to find, together with the relatively slight amount of fracturing to which the older crystals have been subjected, indicates that the crystallization was all accomplished subsequent to the injection."

Dr. J. Lehmann found in Saxony in some pegmatite dikes, orthoclase usually intergrown with albite in form of perthite, and also covered with thick plates of albite. The albite had remained in solution till about the end of the formation of orthoclase.

C. W. Brögger concludes that some albite forms in the pegmatitic dike-veins at a later period than the potash-feldspars, being one of the products of the second period of consolidation, when the chief factor in mineral formation is the influence of the "mineralizers" residual from the previously consolidated portion of the magma. In part this later albite has originated by replacement of earlier orthoclase, showing that the residual solutions had grown sodic.

(P. 129) Under the heading, Development of the Theory of Metalliferous Veins of Magmatic Quartz, Mr. Spurr briefly traces the theories of pegmatite formation through a series of authors from 1823 to 1905. With much further condensation the views are as follows:

Fissure dikes, Charpentier, 1823.

Dike-like segregations, contemporaneous with inclosing rocks, Keilhau in Norway, 1838; Hausmann; G. Kreischer in 1869; G. Woitschach in Germany; Kalkowski; and Teall in England.

Segregation-veins from granitic juice during crystallization of the magma, Scheerer, 1846.

Lateral segregation-veins, T. S. Hunt in Canada, 1863; and H. Credner in Saxony, 1875.

Aqueo-igneous fusion, ranging from jelly-like masses to attenuated solutions, and the results from granite to quartz, Dr. J. Lehmann in Saxony, 1884.

Aqueo-igneous dikes, injected in colloidal state, A. W. Howitt, quartz-veins in Victoria, Australia, 1887.

Intrusive siliceous magmas form pegmatites, and by a series of gradations, differentiations in the same process of magmatic segregations, attenuated solutions may be reached, forming quartz-veins, W. C. Brögger in Norway, 1890.

And about the same opinion as the last, A. C. Lane, 1894.

Normal pegmatites are granitic intrusions, and quartz-veins of the same area are formed by lateral secretion, G. H. Williams, 1895.

"In some cases igneous injection, in some cases aqueo-igneous action, and in other cases pure water cementation, and in still others combinations of two or all of these processes." C. R. Van Hise, 1896 and 1904.

Intrusion of differentiated siliceous magmas aided by water from the wall-rocks, Crosby and Fuller, in New Hampshire, 1897.

Transition from quartz-veins carrying pyrite, argentiferous galena, and gold to feldspar rocks, J. E. Spurr, 1898.

Magmatic differentiation *in situ*, from granitic magmas, F. D. Adams, Ontario granite-veins, 1898.

From pure quartz to pegmatite, Otto Nordenskjöld in the Klondike, 1899; J. F. Kemp, Long Island Sound, 1901; and J. E. Spurr and H. W. Turner at Silver Peak, Nevada.

Tourmaline pegmatite from siliceous differentiation of the magma and tourmaline quartz-veins from further separation into attenuated solutions, G. A. Waller and E. G. Hogg in Tasmania, 1903.

Recognized transitions from granite to quartz-veins, Iddings, Cross, Pirsson, and Washington, 1903.

Gradation from aplite dikes into quartz-veins, A. C. Lawson in California, 1904.

Transition of pegmatite to quartz-veins, J. E. Spurr and G.

H. Garrey in Colorado, and E. C. Andrews in New South Wales; both 1905.

Finally, Mr. Spurr's own views:

(P. 138) "Are very close to those held by Scheerer, Lehmann, Howitt, and Crosby and Fuller."

T. L. Watson,[31] describing the Virginia granites:

(P. 526) ". . . pegmatite, coarse crystallizations of quartz and feldspar with subordinate mica, in dike-like form, abundantly penetrate the finer granites and associated crystalline rocks over much of the Piedmont region."

(P. 536) "They are of granitic mineralogy, without the occurrence of unusual or rare minerals noted in them, and they cut alike the granites and the gneisses.

"They consist of coarse aggregates of feldspar and quartz, with more or less black biotite and a little muscovite. In the Fredericksburg quarries . . . massive granular magnetite and large and small perfect red crystals of garnet are not infrequent constituents."

(P. 538) "Where observed, the pegmatites are sharply defined from the inclosing rock; parallel banding to the walls does not occur; . . . and all of them are entirely massive, without any evidence of pressure metamorphism shown in them."

The following analytical suggestions about crystallization are from leaders of thought on the subject:

C. R. Van Hise,[32] on coarse crystallization in the general mass of the Black Hills granite.

As the granites have been so thoroughly described by Newton and Caswell, not wishing to reiterate he is quite brief:

(P. 230) "They are in the main coarse-grained muscovite-granites, the only important minerals being muscovite, quartz, and feldspar, the latter including orthoclase, microcline, and plagioclase. These granites are sometimes so coarse as to give muscovite approaching that of a merchantable character. These coarse phases are by no means universal; and they pass into rocks which have all the characteristics of muscovite-biotite-granites of the ordinary type."

A. C. Lawson,[33] on the structure of dikes of Rainy Lake region, Canada.

In these dikes, one 150 ft. wide showed feldspars in the center from 4 to 40 times as large as those in the porphyritic zone, 4 ft. from the contact. Generally, the dikes had ophitic struc-

[31] Lithological Characters of the Virginia Granites, *Bulletin of the Geological Society of America*, vol. xvii., pp. 523 to 540 (1906).

[32] Pre-Cambrian Rocks of the Black Hills, *Bulletin of the Geological Society of America*, vol. i., pp. 203 to 244 (1889).

[33] Petrographical Differentiation of Certain Dykes of the Rainy Lake Region, *American Geologist*, vol. vii., No. 3, pp. 153 to 164 (Mar., 1891).

ture intermediate between the porphyritic zone and the granular zone of the center. In the large dike no quartz was observable at the side, some grains were seen at the 4-ft. mark, at 15 ft. they are more abundant, while in the middle it is a most conspicuous ingredient. In a 65-ft. dike, in the ophitic and granular zone—that is, the central half of the dike—the quartz has pegmatitic intergrowth with plagioclase; this dike is of andesitic type, containing appreciable augite and hornblende; this pegmatitic structure also occurred in other dikes with plenty of augite. Although the macroscopic development of quartz was so noticeable in the central half of the dikes, analysis only gave a difference of, say, 2 per cent., from 49 to 51 per cent.; however, in the two largest dikes, 150 ft. and 120 ft. wide, the difference between contact and centers was 10 and 5 per cent., from 47.8 to 57.5 and 52.5 per cent.; the other constituents about balanced. During solidification of the dike the water was driven from the sides towards the center.

A. C. Lawson,[34] describing diabase dikes of the Rainy Lake region, Canada:

(P. 209) "Micropegmatitic quartz is abundant. It is often intimately intergrown with the feldspar, and as the latter is much decomposed, would seem to replace it as a partial pseudomorph, but apatite needles of the same aspect as those which occur as inclusions in feldspar, augite, and quartz, are often seen to be inclosed partly in a feldspar and partly in a quartz grain. The primary origin of the quartz in spite of its pegmatitic character, is however, not beyond doubt."

J. P. Iddings,[35] discussing the liquefaction and crystallization of magmas "which have been found to have diverse laws," and the influence of water-vapor, says that J. W. Judd, following up Dr. Guthrie's experiments, found that dry nitre melted at 320° C. with 29.07 per cent. of water at 97.6° C.

(P. 27) "Inversely, some substances may exist as fluids at temperatures considerably below their ordinary point of solidification by being combined with water; there is a point, however, at which solidification sets in. Mr. J. J. Harris Teall, who has been studying some of Dr. Guthrie's eutectic compounds, finds that upon solidification they separate into their component parts, which interpenetrate one another like quartz and feldspar in granophyre or micropegmatite."

[34] Notes on Some Diabase Dykes of the Rainy Lake Region, *American Geologist*, vol. i., No. 4, pp. 199 to 211 (Apr., 1888).

[35] A Group of Volcanic Rocks from the Tewan Mountains, New Mexico, etc., *ulletin* No. 66, *U. S. Geological Survey* (1890).

(P. 26) "Moreover, if we consider rock magmas as saturated solutions of silicate salts, as Lagorio has done in the paper already alluded to, we may apply to them the law which Sorby deduced for aqueous solutions of salts, namely, that the solubility of those salts, which, like the silicates, expand upon solution and condense upon crystallization, is decreased by increasing pressure. In other words, in such solutions an increase of pressure would tend to crystallize the salts from solution. Hence in a molten rock magma an increase of pressure alone would tend to induce the crystallization of certain silicate minerals from the magma, or might lead to the crystallization of the whole magma."

The descriptions of the pegmatites waver round the same points, and, as G. H. Williams puts it, the modes of formation probably differed too, so that each of the modern observers may be nearly or quite right. The prominent features in the discussion are, the dependence of the mass of the pegmatites on igneous intrusion, or more properly aqueo-igneous, as applied to the influence of water in conjunction with heat in causing the liquidity of granitic magmas; the modification of this fusion in the case of pegmatites by yet more water or aqueous vapors, the additional water having increased with the quartz-content.

Next in importance is the occurrence of accessory minerals, spodumene, beryl, tantalite, phosphates, tourmaline, lepidolite, columbite, cassiterite, garnet, ilmenite, zircon, all of which have been considered closely associated with pneumatolytic action, and most of them classed as mineralizers from the hydrofluoric, boracic, and other acids entering into their composition. For instance:

Review by T. A. Jaggar, Jr.,[36] of the experiments by J. Morozewicz (Warsaw) on synthesis of minerals and volcanic rocks. These experiments consisted of melting suitable compounds in crucibles in a corner of a glass-furnace at a temperature of from 500° to 1,600° C. and examination of the products. Among those experiments:

(P. 306) "Rhyolite and trachyte magmas, with the Al_2O_3 percentage varying from 6 to 20, were fused in large masses under varying conditions of cooling, and for periods of a fortnight or more, solidifying invariably as structureless glass. . . . Finally success was obtained by adding 1 per cent. of tungstic acid to a rhyolite mixture." . . . [There were found under the microscope] ". . . myriads of bipyramidal quartz microlites, . . . hexagonal plates of biotite . . . transparent prisms . . . believed to be sanadine. . . . There had thus been reproduced by 'dry fusion,' with the aid of tungstic acid, an association of the essential minerals of granite—quartz, mica and acid

[36] *Journal of Geology*, vol. vii., No. 3, pp. 300 to 313 (Apr.–May, 1899).

feldspar. . . . Modern petrographers have not ascribed any 'mystical' power to the compounds of tungsten, zirconium, boron, fluorine, etc., but have observed that these elements are minor but invariable accompaniments of the crystallization of coarse acid pegmatites . . . their influence, whether chemical or physical, cannot be denied."

Morozewicz had rather scoffed at the acid having any mystical power.

It has often been explained that exudations from magmas, besides dikes, consist at first of vapors and gases at high temperatures, as seen in fumaroles, which often carry fluorine, and are sometimes as hot as 500° C. at the surface. It is apropos that Brauns has said that a mica can be formed if its elements are melted with a fluoride below 800° C.; above that temperature they are unstable, excepting biotite. The vapors are followed by hot waters, which may originate from them by condensation, either from cooling of the parent magma in depth or in passage of the vapors towards the surface, the seat of origin being sometimes many thousands of feet below it. G. F. Becker thinks some of the Southern Appalachian ore-deposits are now exposed by erosion from 15,000 to 20,000 ft. below their original apex, and other deep-seated deposits are known to be 6,000 ft. below the old surface. There is no telling how much further they extend.

While the minerals deposited by such vaporous action are on the whole different from those due to solfataric waters, yet the two series overlap. Tin- and apatite-veins, and their respective accompanying minerals, are considered representative of the former. As the two types merge into one another (tin is an especially well-known product of ordinary solfatarism), in a district like Cripple Creek, where the veins contain fluorine, a typical representative of pneumatolytic action, and little quartz, speculation could be rife as to what might be found in the veins at great depth. In the pneumatolytic zone, tin, for instance, but it is nipped in the bud by the total absence of this mineral in the superficial areas opened.

As ordinary gold-, silver-, and lead-quartz veins merge into tin and apatite pneumatolytic veins, and both of them into contact deposits, so from the above descriptions of the pegmatites of the eastern United States, I would imagine them to be: eruptive dikes, pneumatolytic veins, and quartz-veins pass-

ing (?) into one another. To be succinct, that the major part of the pegmatites are only ordinarily coarse-grained siliceous dikes, as Crosby and Fuller have outlined, cooling slowly, influenced by a comparatively small amount of accompanying vaporous mineralizers, and that after extravasation and before cooling, pneumatolytic vapors had passed through various channels along the dike, creating the coarse crystallization by keeping certain lengths and breadths of the dike liquid for a sufficient time, allowing further differentiation or formation of eutectic compounds, and that the same pneumatolytic action introduced the rarer minerals.

That after this took place, and after consolidation of the dike, there occurred the next stage of solfatarism, and the introduction in the fissured or brecciated dikes, or in other fissures or lines of least resistance in the adjacent country, of quartz as a residue from attenuated waters.

It may be said that these three phenomena are all phases of the same process of magmatic differentiation; so it may then be granted is the diabase dike on the foot-wall of the Comstock and the lode itself, but they have not been classified together.

C. R. Van Hise[37] has qualified veins and dikes:

". . . the first representing crystallizations from water-solutions; the second, crystallization from magma."

[37] Some Principles Controlling the Deposition of Ores, *Trans.*, xxxi., 288 (1901).

Volcanic Waters.

BY JOHN B. HASTINGS, DENVER, COLO.

(New York Meeting, February, 1908.)

THE origin of the watery vapors of vulcanism has always been an object of interest and speculation to the seismologist, and as theories of the genetic origin of ore-deposits have of late years been pretty well narrowed down to the expiring forces of plutonic action, the same question has attracted mining engineers and geologists, as is well shown by the discussions of the subject in our *Transactions*. The important part taken by volcanic emanations in the origin of pegmatites and quartz-veins, described in my paper, Origin of Pegmatite,[1] and their latent power to concentrate into useful deposits such scattered gold as occurs in the Hartsel granite, make a discussion of their derivation but a natural third and final step.

It is conceded that enormous amounts of vapor accompany vulcanism, though perhaps we are apt to forget that steam has 1,700 times the volume of water, besides which the column seen over Vesuvius and other volcanoes is greatly mixed with air; the immensity of such volumes compared with solids and liquids is shown by the experiment of Gautier, who, by heating dessicated granite to 100° C., evolved from it gases 20 times, and steam 90 times, its own volume. Dana appraised the average amount of water left in ordinary rocks as 2.5 per cent.

T. M. Read estimated that the Mississippi river carries to the sea annually 150,000,000 tons of rock material. If these Mississippi sediments, as deposited, contained 20 per cent. of water, it would be 600,000,000 cu. ft., or 4,500,000,000 American gallons, annually. Allowing the vapor suspended over a volcanic cone to be mixed with 80 per cent. of air, this amount of water converted into steam would replenish anew, every 9 min., a column 15,000 ft. high, 2,000 ft. in diameter at the base, and 10,000 ft. at the top. Read's estimate is only half of the amount given in a more recent and probably correct one.[2]

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[1] P. 104, this volume. [2] Chamberlin and Salisbury, *Geology*, 2d ed., vol. i., p. 106.

In the present paper, however, it is not intended to dispute estimates of the vapors accompanying vulcanism, or what part of it springs from primeval hydrogen and oxygen locked within the earth's interior, or from vadose or oceanic waters. It is simply my purpose to ask whether any of the water buried deeply with oceanic sediments can make its way to the conduits of molten material. An immense amount of water is buried with sediments, and it probably stays there until removed by the rise of the isogeotherms of internal heat—either a gentle rise, or a comparatively sudden one, accompanying movement. Our experience in cyaniding teaches us that water will stay in interstitial spaces until displaced by another substance. Hence, the spaces left in compressed sediments will be full of water, unless the isogeotherms rise as the sediments are laid down and expel it. The gentle action of heat is aided by the expansion and loss of viscosity of warmed water; the latter quality is said to be about one-fifth at 100° C. of what it is at 0° C.

Though enormous pressures can be figured on every square inch of buried sediments, the evidence of compression in them, except where there has been movement, is not great. The pulpy matter of rotted sigillaria, etc., has been crushed to perhaps 1 per cent. of the original volume, and converted into bituminous coal; but even this coal is not a very dense material to preserve its form over areas of many square miles when buried under thousands of feet of rock subsequent to the Carboniferous. It takes extreme pressure and heat to make anthracite.

G. H. Eldridge[3] describes early Tertiary lignite of Cook Inlet, Alaska, many of the beds being from 4 to 6 ft. thick, as:

". . . hardly more than a compressed mass of carbonized wood, it being possible to pull up from the back of a seam slivers from a few inches to 3 feet in length. Stumps 1 foot to 2 feet in diameter are common . . . Indeed, in some of the local bogs resting upon the coal measures there are stems and other woody tissues that closely approximate some of the less altered varieties of the lignitic material."

I do not know how thick the superincumbent beds over the coal were originally. W. H. Dall,[4] who studied the region,

[3] A Reconnaisance in the Sushitna Basin and Adjacent Territory, Alaska, in 1898, 20*th Annual Report, U. S. Geological Survey,* pt. vii., pp. 7 to 29 (1898–99).

[4] Report on Coal and Lignite of Alaska, 17*th Annual Report, U. S. Geological Survey,* pt. i., pp. 771 to 875 (1895–96).

says that at this particular point these Tertiary Kenai coal-beds are conformably overlain by Miocene marine beds, but the subsidence which enabled it was "probably moderate in vertical range."

As to the rise of the internal isogeotherms from normal conductivity or pressure, there is little light, except theoretical. The continents show increased temperature with depth, but they are areas of considerable movement—elevation—which in itself would be accompanied with heat, and what remains may be, in part, residual. The same might be said of the ocean bottoms, substituting a movement of depression.

If the heat-conductivity in the earth be judged by its radiation it is very slight. It has been described as small, but still measurable. The abyssal ocean waters are just above freezing-point. This low temperature has been thought to be due to rapid extraction of the earth's heat from the oceanic beds by conductivity because of their greater density than the continents, for instance, and the rapidity with which the waters absorb the heat and quickly distribute it through convection. Again, it has been ascribed to the immense polar currents moving slowly southward along the bottom. It is hard to find any warmth in the ocean except that due to the rays of the sun. Chamberlin and Salisbury,[5] citing P. G. Tait's *Heat* and A. Daniell's *Physics*, assert that these authors respectively place the earth's loss of heat in 100,000,000 years at 18° F. and 81° F. for the whole body of the planet.

Do the sediments as deposited contain much water, and do interstitial spaces survive compression of the beds?

J. D. Dana thought that

"sedimentary beds contain their maximum of moisture when laid down; and with this, though situated at the bottom of a trough, they have still laid quietly."

Not much has been said of late by geologists about water carried down in sedimentary beds as they are deposited. Sterry Hunt and others referred to it, and thought that some eruptive rocks were formed from saturated sedimentaries.

Perhaps the only emphatic recent article bearing on the subject is by Professor King,[6] and not only treats of the amount of

[5] Chamberlin and Salisbury, *Geology*, 2d ed., p. 572.

[6] F. H. King, Principles and Conditions of the Movements of Ground Water, *U. S. Geological Survey*, 19*th Annual Report*, pt. ii., pp. 59 to 294 (1897–98).

water the sediments might once have held, but presents a study of them after elevation, and shows their storage-capacity up to the present time. The following passages are quoted from it:

(P. 69) ". . . sandstones lying below drainage outlets may contain as high as 38 per cent. of their volume of water. . . The Dakota sandstone, for example, stretching from the foothills of the Rocky Mountains eastward beneath the plains of the two Dakotas, Nebraska, and Kansas, apparently in one nearly or quite continuous sheet, may be likened to a submerged inland sea or lake, for wherever this formation lies beneath the zone of saturation it carries within itself from 15 to 38 feet of water on the level for every 100 feet in thickness of the sandstone itself, and from it water may be drawn wherever it lies close enough to the surface to be reached by wells. The Potsdam sandstone is a formation of much wider distribution than the Dakota, and in southern Minnesota and Wisconsin and in Illinois and Iowa it has a measured thickness of 500 to 1,000 feet, all lying beneath the surface of saturation, so that in this great bed there has been stored away a quantity of water equal to a sheet not less than 10 to 38 feet in depth for each 100 feet in thickness, and 500 feet of this water-bearing rock may store the equivalent of an inland submerged sea having a mean depth of 50 to 190 feet of water."

Ordinary clay with sand held 25 per cent. of water, and very fine sand 17 per cent. Marble held very little, 0.25 per cent., or a column of 5,000 ft. held 30 ft. of water.

(P. 77) "We know that when sediments are laid down on the borders of the ocean or over the bottom of inland seas, gulfs, or bays there becomes locked up with them large volumes of water, quantities varying from 25 to 50 per cent. of the volume of the sediment, according as the pore space in the sediment is large or small.

"It will be seen in another part of this paper that the pore space of loose sands, when packed as closely as tamping and jarring will secure, amounts to from 30 to 38 per cent., while the pore space of clays and finer soils runs up as high as 45 and even above 50 per cent., and there is no reason to suppose that the sediments, whether they be sands, clays, or limestone débris, will be laid down with greater compactness than we have been able to secure in our experimental work.

"Seelheim[7] has shown that when an emulsion of fine clay and water is allowed to stand quietly for some time under conditions where no jarring can take place the clay subsides, assuming a stratified condition, but containing a large amount of water. He found where no jarring took place that the upper layers contained more water than the lower ones, the proportion being 1 volume of clay to 3.84 volumes of water in the upper strata and 1 volume of clay to 1.78 volumes of water in the lower strata. That is to say, in the loosest settling 79.34 per cent. of the volume of the sediment was water and in the closer packing there was still 64.03 per cent. of pore space.

"Where the settling was allowed to take place under frequent jarrings Seelheim secured a uniform texture throughout and greater compactness, but there was

[7] Methoden zur Bestimmung der Durchlässigkeit des Bodens, *Zeitschrift für analytische Chemie*, vol. xix., p. 387.

still a pore space of 54.54 per cent. He further showed that there was no sensible reduction of pore space when the sedimentation was caused to take place under a pressure of 102 feet of water instead of a few feet. Further than this, there is no reason to suppose that the pore space of sediments laid down under water will not be filled very largely with the water in which they are deposited."

Experiments with sediments laid down in a cylinder showed that the water flowed downward or laterally through the sediments and up through a rubber tube discharging 6 in. above the top of the column of water and sediments; it was not denied that some water, as the sediments were deposited, may have come up vertically through them, but the lateral and downward outlet seemed the line of least resistance.

(P. 80) "We have no quantitative measure of the amount of compression which, under the conditions of natural sedimentation, takes place where beds of shale and limestone are formed. . . . .

" . . . Such consolidation of sediments and displacement of water as may have taken place by the ordinary processes of sedimentation unaided by other agencies must still have left large volumes of inclosed water to be deeply buried in districts like the Appalachian region, where, during Palæozoic time, if the estimates of Dana are accepted, an aggregate subsidence and sedimentation of 36,000 feet must have taken place."

When these materials were laid down the pore-space was about 33 per cent., or 12,000 ft. of water. But these rocks have since been greatly consolidated, highly metamorphosed and crystallized, perhaps making the pore-space much less than 33 per cent., so that if they were now full of water a very large amount would have been displaced in the past. The same thing has occurred in the interior region of our continent.

(P. 81) "These large volumes of water which have been carried beneath the earth's crust as a phase of the process of sedimentation must in part have reappeared at the surface in one place or in another, and there must of necessity be an underflow of the entrapped sedimentary waters from beneath the ocean toward the land." [Such subsidence produces increased temperature of the material, and resultant flow of water.—J. B. H.] "Poiseuille found that water at a temperature of 45° C. flowed 2.5 times as fast under otherwise like conditions as water at 5° C."

G. F. Becker[8] says quartz weighs 165 lb. per cu. ft., and resultant sand well shaken down 120 lb. per cu. ft.; therefore, the sand contains 27.3 per cent. of interstitial space. This sand

[8] Geology of the Quicksilver Deposits of the Pacific Slope, *Monograph No.* 13, *U. S. Geological Survey*, p. 399 (1888).

compared very well with the natural sand-beds and sandstones of the California quicksilver-mines.

"Were the sand composed of spherical grains all of the same size and as closely packed as possible, so that every sphere was in contact with twelve others, the mass would contain 26 per cent. of interstitial space."

Indurated sandstone used for paving the streets contains 8 per cent. of interstitial space.

W. O. Crosby,[9] in discussing the coloration of the sandstones near Colorado Springs, inadvertently suggests limitation of inclosing water:

"In deeper and more quiet water offshore, where the overlying reddish sandstone must have been deposited, enough red clay would also naturally have been deposited to fill the interstices between the grains and give the sandstone a ruddy tint."

On this principle, it is somewhere stated that conglomerates may contain as low as 7 per cent. of water.

The results of a series of experiments on the porosity of rocks, made both by himself in Canada and by others in England and France, are tabulated by Sterry Hunt.[10]

The only reference I can find as to what actually became of the originally inclosed sedimentary waters, except those consumed in chemical hydration, is the mention of saline springs in eastern United States and Canada, with analyses of them, by Sterry Hunt, in his essays.

Do sediments subside until they are an immense depth below the ocean bottoms?

This is the general opinion of geologists.

J. W. Dawson[11] wrote that from 10,000 to 20,000 ft. of sediments in Nova Scotia and eastern United States were laid down in shallow seas—that is, the bottom of the seas subsided as they were deposited.

In the Alps the sediments accumulated between the Permian and late Cretaceous periods are thought to exceed 50,000 ft., all accumulated in shallow water. Dana says:[12]

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[9] Archean-Cambrian Contact Near Manitou, Colorado, *Bulletin of the Geological Society of America*, vol. x., p. 164 (1898).

[10] *Chemical and Geological Essays*, 2d ed., p. 165 (1878).

[11] *Some Salient Points in the Science of the Earth.*

[12] *Manual of Geology*, 3d ed., p. 391.

"Consequently, when these last layers of the Palæozoic in the Appalachian regions were at the ocean's level, the Potsdam beds—though once also at the surface—were about *seven* miles below; for this is the thickness of the strata that intervene; seven miles of subsidence had, therefore, taken place in that region during the progress of the Palæozoic ages."

Clarence King judged that the subsidence in the Rockies amounted to 60,000 feet.

S. F. Emmons[13] thought that from early Cambrian to early Carboniferous the great accumulation of beds in the longitude of the Wasatch mountains amounted to from 15,000 to 20,000 ft., not strictly without movement of elevation and depression, but without great disturbance.

Bailey Willis says:[14]

"In the Appalachian trough opportunity for maximum sedimentation during the Paleozoic was afforded by the profound subsidence of an area in New York, Pennsylvania, Virginia, and Ohio. At Mauch Chunk the total thickness is approximately 30,000 feet."

R. T. Hill,[15] describing the Beaumont oil-fields, refers to 25,000 ft. of marine sediments, from Cambrian to recent, minus the Devonian, and says they are—

"tilted steeply in the mountainous areas, and are nearly horizontal in the plains."

(P. 400) "Of the 22,000 ft. of sedimentaries in the Texas section, all but less than 2,000 ft. are unconsolidated clays and sands."

These examples might be multiplied. Chamberlin and Salisbury, in their beautiful new *Geology*, give a touch of the other side of the question. They allow that subsidence has gone on in epicontinental (shallow bordering) and mediterranean (deep inland) seas, the inland sea of North America, for instance, and do not deny that it occurs elsewhere, but think that errors in computation may have been sometimes made. For instance, suppose sediments deposited conformably on the abyssal floor of an ocean, which slopes 2° until a depth of 4 miles is reached, whence the floor goes off flat, but the sediments keep on piling up conformably at 2° dip till they are built out 200 miles from

[13] Geology of the Denver Basin in Colorado, *Monograph No. 27, U. S. Geological Survey*, p. 15 (1896).

[14] A Theory of Continental Structure Applied to North America, *Bulletin of the Geological Society of America*, vol. xviii., p. 399 (1906).

[15] The Beaumont Oil-Fields, etc., *Trans.*, xxxiii., 368 (1903).

shore. An observer at the shore end, taking the angle of dip of the first bed laid down, and protracting it for the 200 miles, which on this dip would take it away below the ocean bottom, and then calculating the distance at right angles to the dip from the bottom of this first sediment to the top of the last one laid down, would get 7 miles, instead of 4, the real depth of the ocean and true thickness of the sediments. It is certainly plausible that protracting a dip for 200 miles might lead to error. If it were protracted 5 miles the discrepancy would be 0.15 mile. Another citation is given of a lake 100 miles wide, with sides sloping at 3° till a depth of 1,000 ft. was reached, whence it remained flat. The sediments were then laid down conformably at an angle of 3°. The observer takes the angle of dip at the shore and protracts it at 3° for 50 miles, getting so far below the bottom of the 1,000-ft. lake that his computation at right angles to the 3° dip, at end of the 50 miles, gives a depth of 13,800 ft., instead of 1,000 ft. These layers of sediment, diagonal to the depth of the lake, would butt against its bottom in about 3 miles—that is, if any of the beds were followed from the outcrop down the 3° dip for that distance, they would be found ending in a truncated edge against the lake bottom, so that their horizontal length would be little more than the supposed thickness, whereas the Appalachian relations, as computed, are in horizontal extent 20 times or more their thickness.

Are the continental shores lines of weakness, so that there is a possibility of the inclosed waters of the sediments passing from them into the land areas?

While there are certain parts of the present continents and also oceanic depths which are thought to have existed since the dawn of geologic history, it is well known that almost all of the inner area of North America has been occupied by a mediterranean, so that tremendous thicknesses of sediments have been laid down, in Arizona, Utah, Idaho, and British Columbia; that the ancient sea of Tethys stretched from Spain to the eastern coast of Asia; that the Arctic was once much larger than now, and also that the southern continents, Africa, Australia, and South America, were apparently closely connected. These facts have been worked out with much research by such men as Dawson, Dana, etc., on our own shores, and by others elsewhere.

The facts have been lately collated by Bailey Willis, who says, in effect,[16] that uplands of erosion and lowlands of aggradation are commonly joined by a monoclinal flexure, or a normal fault. Great horizontal movements are shown in the schistose structure of once deep-seated rocks. These movements crowded the continental elements together, as is evidenced by the shortening of the Appalachian sediments from their original horizontal breadth; according to the Rogerses, from 168 miles to 60 miles. Lately, however, the narrowing has been put at 35 to 45 miles. Much may depend, perhaps, on the locality of the section observed.

That oceanic areas might be denser than continental, probably dawned early in a number of minds and was mentioned by some, but Captain E. P. Dutton was the first to formulate definitely the theory of "isostacy," which predicates that the sectors of the oceans and continents are in equilibrium, the depressions of the one and elevations of the other equalizing the specific gravity of the two.

Willis calls the oceans and continents positive and negative elements, and specifies the line of contact between them as a zone of weakness, especially favorable for the development of intrusive and extrusive bodies.

It may be interesting for a moment to glance at B. K. Emerson's[17] statement of the theory of a tetrahedral earth as postulated by its originator W. L. Green, a Honolulu merchant:

"The sphere, of all solids, contains the greatest volume under a given surface, the tetrahedron the least volume under the same surface. The solid spherical crust of the earth, then, collapsing upon its plastic interior, would tend toward the tetrahedral form as the one which would coördinate the greatest diminution of the interior with the least change of the surface."

The continents are shown to be situated more or less on the protuberances of the tetrahedron. Emerson says the tendency for the earth to take such a form may be allowed to exist and would be in evidence where not balanced by other forces.

However approached by geologic theory, I think it is quite generally acknowledged that continental shores are broad lines of weakness. Furthermore, that the general arrangement of

[16] The Mechanics of Appalachian Structure, 13*th Annual Report, U. S. Geological Survey*, pt. ii., pp. 211 to 281 (1891–92).

[17] The Tetrahedral Earth and Zone of the Intercontinental Seas, *Bulletin of the Geological Society of America*, vol. xi., pp. 61 to 106 (1899).

volcanoes, as taught by Dana and others, is symmetrical with these lines. This does not preclude volcanic action far away from the coast. If the elevation of the continents is due to the thrust of the denser oceanic beds large distances may be insignificant. Neither does it preclude vulcanism under the ocean. In fact, J. W. Dawson has described the Atlantic as "almost an unbroken cake" and the Pacific as "cracked in many places, allowing the fluid matter to exude in volcanic ejections." Nor does it matter that there are quiescent areas, where under this law vulcanism might be expected to be active. Many have observed the marked feature of critical epochs in the world-history, that there have been geological periods of slow subsidence and elevation without much vulcanism.

What is the depth of vulcanism?

This may be great indeed, but the depth of earthquake centrums, which we associate with movements in the earth's crust (faulting), originally considered as great (even from 30 to 55 miles by Mallet), has of late years been restricted to 5, 8 or 12 miles, which brings it not so much below the sedimental range.

Are the waters of vulcanism similar to those included in sediments?

Since it is not supposed that these waters are exclusively derived from the sediments, they may be greatly modified mixtures as we see them. Prestwich says:[18]

"Not only are almost all the elements of sea-water found in the gases and deposits of fumaroles, with the exception of the salts of magnesia which has become fixed in the lava, but sea-salt itself is often found in lava or is deposited on its surface."

The *Challenger* reports give the salts of sea-water as roughly: sodium chloride, 78; magnesium chloride, 11; magnesium sulphate, 5; calcium sulphate, 4; potassium sulphate, 2; calcium carbonate, 0.25; magnesium bromide, 0.25 per cent. The total quantity of salts is fairly uniform as 3.5 per cent. of the water. The salts vary; sometimes the bromides are more plentiful, and strontium, iodine, fluorine, glauconite, etc., are found.

Are oceanic sediments usually dehydrated before or during elevation, and which way is the water driven, upwards or laterally, and if laterally, seawards or landwards, and if landwards, is it likely sometimes to penetrate, and incorporate with, plutonic magmas, and at depths beyond any it is agreed vadose waters can reach?

[18] *Geology: Chemical, Physical and Stratigraphical*, vol. i., p. 213 (1886).

Genesis of the Lake Valley, New Mexico, Silver-Deposits.

BY CHARLES R. KEYES, SOCORRO, N. M.

(New York Meeting, February, 1908.)

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I. Introductory.

Lake Valley, New Mexico, has long been one of the most widely known mining districts of southwestern United States. For many years its silver-mines have been among the most famous of the country, visited by many mining men and geologists; yet little information regarding the origin of the ore-deposits and the influencing geologic conditions has found its way into print.

With the recent revival of public attention, after a decade and a half of utter stagnation, to the mining of silver in the Southwest, special interest attaches to the genesis and geologic disposition of the ore-bodies which are so finely open to inspection in the Lake Valley district. Moreover, the local features here described are typical of the conditions existing over a much larger field. The fact that the ores are chiefly the chloride and the chloro-bromide of silver adds further interest. Data bearing upon the genesis of haloid ore-deposits are, at the present time, very much sought. Arid regions offer particularly favorable conditions for the formation of such ores. This New Mexican district appears to supply solutions to many problems concerning this class of ores, which have long vexed the student of ore-deposits.

II. Location and History.

The town of Lake Valley is situated in Sierra county, in the southwestern part of New Mexico, and 100 miles northwest of El Paso, as the crow flies.

From a mining standpoint, the location of the place is itself suggestive, as it lies on the southeastern slope of the great quaquaversal arch of the Colorado plateau. The center of this vast dome is in north-central Arizona, and radii about 150 miles in length reach its foot in all directions. Around its basal margins runs the broad mineralized belt containing most of the great mines of the Southwest, which extends from south-central Colorado southward and southwestward through New Mexico, thence westward through Arizona, northwestward and northward through southeastern California and western Nevada, and, finally, eastward through Nevada and Utah into Colorado again. This great mineralized belt is important as geographically limiting profitable mining over the greater part of five States.

Beyond its confines, mining is to be indulged in with the utmost caution.

The recognition, upon tectonic grounds, of the great ore-bearing zone should have a direct and far-reaching influence upon the systematic exploration for mineral products throughout the southwestern United States.

The history of the camp of Lake Valley need not be here narrated, except so far as to correct some erroneous statements of long standing, and to connect the names of several distinguished mining men and scientists with the various enterprises. The discovery of silver in the district was made in 1876 by Mr. McEverts, a cattle-man, who located several claims, which afterwards yielded some of the richest ores. Two years later, while stopping at the McEverts ranch, Mr. George W. Lufkin, a civil engineer from New Jersey, saw specimens of the high-grade silver-ores, and immediately acquired a half-interest in the property. From this time on the development of the camp was rapid. Among the eminent names connected with the various enterprises of the district, at one time or another, may be mentioned those of George Lufkin, George D. Perkins and Whittaker Wright, of Philadelphia; Dr. F. M. Endlich and Prof. E. D. Cope, scientists of the Hayden governmental surveys; Ellis Clark (who has left us the only succinct account of the mine-operations), E. W. Hadley, and S. A. Miller and Frank Springer, who first determined the exact geological age of the country-rocks.

The fickleness of fortune is exemplified in the case of Lufkin, who, after sinking a shallow shaft, sold out his interests for a song. Before his successors had pushed the work in the shaft 2 ft. deeper they struck the "Bridal Chamber," a small pocket of very high grade ore, which yielded more than $1,000,000.

III. Surface-Relief.

The rugged topography of the region immediately about Lake Valley derives its characteristic features from the differential weathering of strongly-tilted rock-layers under unusual conditions of climate, and the vigorous activity of geologic agencies unknown in more humid regions. The alternation of thick beds of very resistant limestones, or of eruptive sheets, and soft shales, in faulted and highly inclined strata, has given

to the relief extreme and unfamiliar characters, the most striking of which are series of long, deep, sharp-bottomed, parallel valleys, worn out of the thick shale-beds, and separated by high, serrated ridges made by the harder layers. This is probably the main reason that the rich silver-ores lying on the surface escaped the notice of the early Spanish explorers and adventurers, although the locality was less than half a dozen miles from one of the principal trails, traveled for more than three centuries. For the location of the ore-deposits, in the bottom of a deep, secluded valley, surrounded on all sides by mountains and hills, and with no marked drainage-ways leading into it, naturally tended to leave them to be discovered by mere accident, rather than by intelligent search. This was, in fact, what happened. The first location was made, not by a mining man or even a prospector, but by a *ranchero* of the neighborhood, who afterwards learned from others the value of what he had thus acquired.

The peculiarities of the surface-relief and of the degradation of the district thereby conditioned, have had, as will be seen, an important effect in the segregation of the ore-bodies. The main drainage is northward through the parallel valleys and with the strike of the rocks. In very recent times a change has begun to take place in the eastern part of the area, and narrow dip-valleys have been developing through head-water erosion, so that now the storm-waters of the mining belt are largely carried out through the new channels instead of the old strike-valleys. As explained below, the economic effect of this drainage-change has been to permit the escape of most of the impounded underground waters of the mining belt, together with the mineral, as fast as it was converted into soluble forms. Instead of being merely deposited a little farther down the slope, as had been the case since the time of the Berenda faulting, the greater part of the original ore-body has been, in this way, gradually lost.

IV. General Geological Features.

1. *Main Relationships and Map.*—In the consideration of so small a district, its relationships with those of the great Colorado dome are almost completely lost. Moreover, in so limited an area the most striking local geotectonic feature is

really the block-mountain, on a diminutive scale. The main geologic characteristics of the district are indicated on the accompanying sketch-map, Fig. 1.

2. *Résumé of Geologic History.*—The later geologic history of the district, which has to do directly with the formation of the ore-deposits, may be briefly summarized as follows:

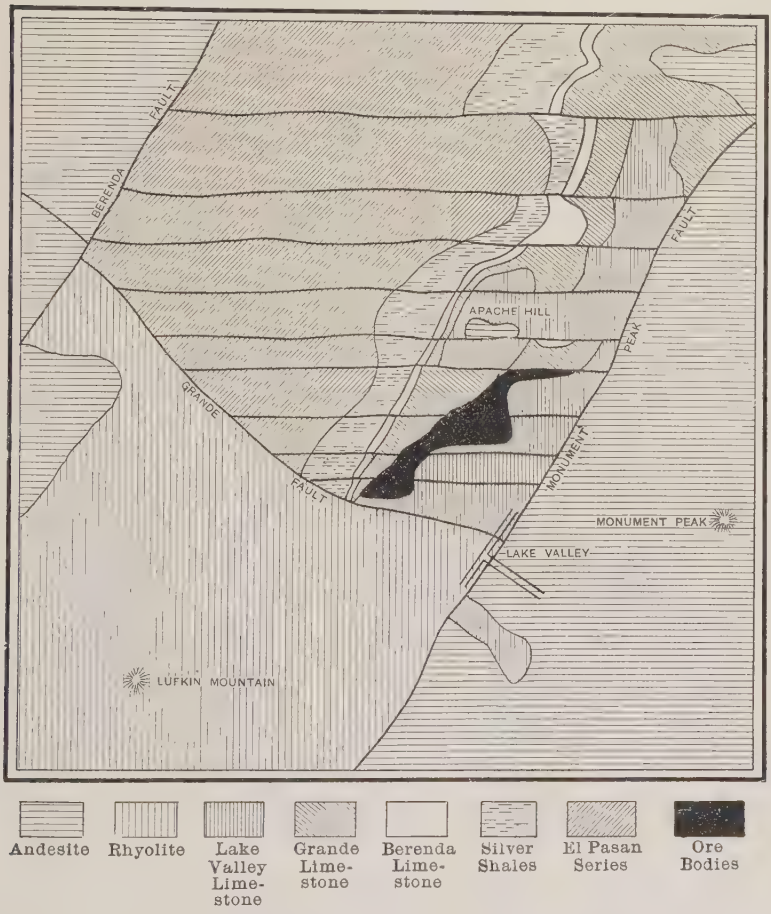


FIG. 1.—GEOLOGICAL SKETCH-MAP OF THE LAKE VALLEY DISTRICT, SHOWING FAULTS.

(1) Rhyolitic eruptions, accompanied by important local mineralization of the bedded rocks at and near the contacts.

(2) The "Grande" faulting, profound and trending northwest and southeast; much more marked, however, beyond the limits of the district.

(3) Vigorous erosion, the most obvious effect of which upon the local surface-relief was marked planation.

(4) Andesitic outpourings, not all confined to a single period, resulting mostly in comparatively thin sheets, but covering a wide scope of country, and almost entirely hiding the stratified rocks.

(5) The "Berenda" faulting, trending northeast and southwest, causing a marked tilting of the rocks, bringing again to the surface the old Paleozoics, and inaugurating local conditions highly favorable to the segregation of ore-materials.

(6) Marked erosion, greatly accentuating through differential effects the local geologic structure and aiding the formation of important ore-bodies.

(7) The "Bella" faulting, consisting of minor, closely-grouped parallel dislocations trending east and west, and often sharply limiting ore-bodies.

(8) Modern erosion, affecting ore-deposition, first, by notable localization, and then by notable impoverishment.

The events thus enumerated in sequence are only those which have been most intimately connected with the local segregation of the ore-bodies. They are all subsequent to the formation of the Colorado dome, and the main epochs of ore-concentration around its base. The full consideration of the last-mentioned event belongs more properly to a broader treatment of the regional geology than to a local description.

V. Geologic Formations Represented.

1. *Broader Considerations.*—The base of the southeastern slope of the Colorado dome is of special interest stratigraphically, because it is a sharply-defined line, separating a vast area, from which the entire Paleozoic section is absent, from another immense region, throughout which the Paleozoics are very fully represented. Within the boundaries of the first-mentioned more northerly area, only the two uppermost Paleozoic terranes are represented. Middle-Carboniferous limestones rest directly upon the oldest Azoic gneisses and schists. In the more southerly area, every system of the Paleozoic section is present. In the northern region of New Mexico, these Carboniferous formations are the chief ore-carriers, while in the southern region, the ores are found through several thousand

feet of Paleozoic rocks, wholly unrepresented in the north. In every way the two regions are strongly contrasted.

Lake Valley is on the border line just described. Hence, its geologic section has both a mining and a geologic importance which it would not otherwise possess. It is also especially instructive since it is, in almost every respect, the exact counterpart of the standard Devono-Carboniferous section of the Mississippi valley. The recent recognition of this resemblance has been fully considered in another place.[1]

The definite determination of the geologic age of the different rock-formations in which the ore-deposits of this region are found acquires practical value by enabling the search for ores to be quickly narrowed to limited horizons. This fact is strikingly illustrated in the neighboring districts, where the rocks have been repeatedly and profoundly faulted, and otherwise disturbed. But, even in the Lake Valley district, notwithstanding the clearness with which the geologic formations are displayed, a large amount of unwise and futile prospecting-work has been done.

2. *Geologic Section.*—The high tilting of the rock-layers gives within the distance of less than a mile an exposed section, complete without a break, as shown in Table I.

TABLE I.—*Geologic Section at Lake Valley.*

	Age.	Formations.	Thickness.	Rocks.
Cenozoic.	Quaternary.		100 ft.	Gravels and Loams.
	Tertiary.	Monument. Lufkin.	250 ft. 500 ft.	Andesites. Rhyolites.
Paleozoic.	Carboniferous.	Sierra. Lake Valley. Grande.	50 ft. 150 ft. 25 ft.	Gray Limestones. Shaly Limestones. Blue Limestones.
	Devonian.	Berenda. Bella. Silver.	50 ft. 60 ft. 100 ft.	Nodular Limestones. Green Shales. Black Shales.
	Silurian.	Santa Rita.	10 ft.	Drab Limestones.
	Ordovician.	El Pasan Series.	600 ft.	Dark Limestones.

[1] *Proceedings of the Iowa Academy of Sciences*, vol. xiii., pp. 197, 198 (1906).

3. *Nomenclature.*—Some of the geographic titles of the formations have already come to have a much wider than local application, for which reason it is thought advisable to retain, not only these, but all of the local names. A further reason is that the Lake Valley section is typical for the region, and that the several terranes were here first recognized and described. The geologic ages of the several formations have recently been attested by extensive collections of fossils.

4. *El Pasan Limestones.*—The El Pasan series[2] of limestones, so well displayed in the Franklin mountains north of El Paso, is exposed at Lake Valley with more than 600 ft. thickness, and is doubtless much thicker. The organic remains found in it appear to be the same forms as occur abundantly 100 miles to the southeast. Little doubt is entertained that the rocks are Ordovician in age. The series appears to comprise several well-marked subdivisions. Detailed faunal inquiries will soon delimit clearly the different members. In many localities in the Southwest, these limestones cannot be lithologically distinguished from the overlying Devonian and Carboniferous blue and gray limestones. When the dark-colored Devonian shales are absent, as they often are, the limestones form an unbroken succession of great thickness.

The limestone range west of the mines is called the Quartzite ridge. This title is misleading. The quartzitic aspect is occasioned by large masses of weathered flint, which extensively cover the surface. Since no part of the series carries any ores in this locality, and its subdivision into minor members is unimportant in the present connection, it demands no further description here.

5. *Santa Rita Limestone.*—Exposed in shafts sunk to the Ordovician formations there is found, immediately overlying the latter, a dark compact lime-rock, about 10 ft. thick. The crevices, cracks and uneven places in the surface are filled with a fine, bright red clay, appearing at first glance like a peculiar reddish lime-rock. The formation is doubtless very much thicker elsewhere in the neighborhood; and it is also absent entirely in other places, owing to the marked unconformity, due to erosion, which exists between it and the later terranes.

[2] See *American Journal of Science*, Fourth Series, vol. xxi., No. 124, p. 298 (Apr., 1906).

This limestone is probably a remnant of the true Silurian formations, exposed a few miles to the west, and also to the southeast, of Lake Valley. The name Santa Rita has been suggested for the formation. In the vicinity of Santa Rita and Silver City, 30 miles west of Lake Valley, abundant fossils found in apparently the same terrane suggest typical Silurian forms. Still farther west, in eastern Arizona, Reagan[3] has discovered a similar lime-rock, 70 ft. thick.

6. *Silver Shales.*—This remarkable bed of black argillaceous shales appears to form the base of the Devonian section of the region. Thus far it has proved to be non-fossiliferous. Its age is deduced from its stratigraphic position; and it is believed to be the eastward extension of a similar black shale found near Silver City, which has been called Silver shales—a name which is here retained.

Although the Silver shales have no direct relationship with the Lake Valley ore-bodies, they have been largely prospected, and many shafts have penetrated these strata.

7. *Bella Shales.*—The green shales so well exposed in many outcrops have been encountered also in the shafts of the Bella workings. Their lithologic characters are the same as those of the black shales beneath. They do not appear to carry recognizable organic remains at any point in the immediate vicinity of Lake Valley. The geologic position and character of the two great shale beds are significant.

8. *Berenda Limestone.*—The so-called nodular limestone of this district immediately underlies the principal ore-bearing formation, with a thickness of about 50 ft., and comprises three distinct and easily recognizable members.

The lowest is a limestone, 10 ft. thick, gray, massive, compact, and resembling somewhat lithographic stone. It weathers rusty brown and is then somewhat laminated. It carries a typical Late Devonian fauna.

The median member, about 30 ft. thick, is gray, thinly-bedded, apparently without organic remains of any kind, and presents in individual layers only the lithographic texture. It weathers irregularly, having on exposed surfaces a notably nodular appearance. There is little or no chert in it.

[3] *American Geologist*, vol. xxxii., No. 5, p. 278 (November, 1903).

The upper member is not more than 10 ft. thick, bluish in color, rather thinly-bedded, composed very largely of chert in elongated nodules and nodular bands, and, so far as known, without fossils. This member contrasts strongly with the massive compact blue limestones lying immediately above.

The entire Berenda formation is also known locally under a number of other titles.

9. *Grande Limestone.*—All the Lake Valley silver-ores are found in this formation. They were first opened at the Sierra Grande workings, and are confined chiefly to the upper part of this tri-partite terrane.

At the base of the formation is a massive, hard, grayish-blue coralline lime-rock, from 13 to 15 ft. thick. Above this is 2 ft. of black, sub-crystalline limestone, carrying abundant large corals and gastropods. The upper member, 10 ft. thick, consists of blue, compact, heavily-bedded, pure, soluble lime-rock, with but few fossils.

Above permanent water-level this formation is usually more or less cavernous. It is at this level that the ores are chiefly segregated. Much massive chert is also accumulated in the old caverns and crevices.

The Grande limestone terrane appears to correspond to the Chouteau limestone of the standard Mississippi Valley section. Special attention has been recently called to this fact;[4] and the details need not be repeated here.

10. *Lake Valley Limestone.*—The geologic age of this Early Carboniferous formation was first determined in 1881, by Mr. S. A. Miller,[5] from fossils collected by Prof. E. D. Cope, of Philadelphia, who was at that time President of three of the Lake Valley mining companies. More extensive collections of fossils were made several years later by Mr. Frank Springer,[6] who recognized the fauna as corresponding to that of the Lower Burlington limestone of Iowa, a formation which has been, of late, widely recognized in New Mexico.[7]

At Lake Valley this formation is 150 ft. thick, and consists chiefly of more or less shaly and highly fossiliferous limestones.

[4] *Proceedings of the Iowa Academy of Sciences*, vol. xiii., pp. 197, 198 (1906).

[5] *Journal of the Cincinnati Society of Natural History*, vol. iv., pp. 306 to 315 (1881).

[6] *American Journal of Science*, Third Series, vol. xxvii., No. 158, p. 102 (Feb., 1884).

[7] *Proceedings of the Iowa Academy of Sciences*, vol. xii., pp. 169 to 171 (1904).

The Early Carboniferous rocks of New Mexico are regarded as belonging to the Socorran series,[8] corresponding to the Mississippian series of the Interior Basin region.

11. *Sierra Limestone.*—The uppermost member of the clastic series at Lake Valley differs very materially from all other terranes in being a gray, compact, massively bedded lime-rock, of distinctly coralline character, yet exhibiting but few well-preserved organic remains. The fossils thus far found in this upper limestone suggest strongly the fauna of the Upper Burlington and Lower Keokuk limestones of the Mississippi valley. The formation has an exposed thickness of about 50 ft.

VI. Unconformities.

The Lake Valley section includes three horizons at which occur marked breaks in the sedimentation. All three of these planes display discordance between the strata, and hence represent periods of erosion. While these have no doubt affected the ore-deposits of Lake Valley, they have had, in the neighboring districts, a much greater influence upon the distribution of the ores.

The most marked plane of unconformity is at the base of the black Silver shales. The direct effects of this pre-Devonian erosion were far-reaching. Over most of the region the Silurian rocks, together with much of the Ordovician formations, were entirely removed. In some places, however, remnants of the Silurian were preserved.

A second unconformity occurs at the base of the Carboniferous limestones. The evidence of this discordance in sedimentation is not so apparent at Lake Valley as elsewhere in southwestern New Mexico. Nevertheless, the shales beneath the Blue limestone have been, in many instances, entirely removed, so that this lime-rock rests directly upon Silurian or Ordovician limestones of similar lithologic character.

The third marked unconformity is shown by the eruptive rocks lying upon the beveled edges of the sedimentaries. In connection with these, some later clastic formations also occur.

[8] *Journal of Geology*, vol. xiv., No. 2, p. 151 (Feb.–Mar., 1906).

VII. Geologic Structure.

1. *Main Characteristics.*—Viewed in its entirety, the tectonic aspect of the Lake Valley district is that of a monoclinal block tilted about 20° southeastward, in which mass, thus disturbed, differential weathering has made deep valleys of the non-resistant formations, and long, sharp, parallel ridges of the hard beds, as explained above. This structure is diagrammatically represented by the cross-section shown in Fig. 2.

2. *Folding.*—Prior to the period of the Berenda faulting, which produced the main monoclinal block, the most notable structural feature was a series of gentle folds. This undulatory character of the strata is still plainly preserved, in spite of frequent subsequent faultings and other marked deformations. The direction of the compressive forces was northeast and southwest.

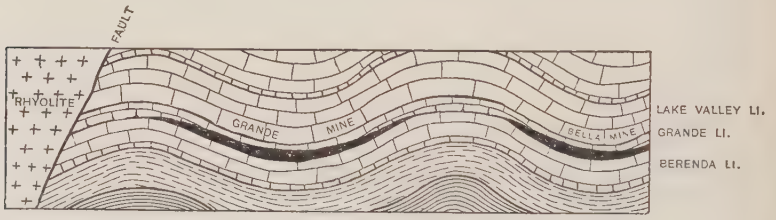


Fig. 2.—Cross-Section of Synclines, with Ore-Bodies in Bottoms.

In the subsequent southeastward tilting to which the beds were subjected, the inclination was in the direction of the axes of the great corrugations. It was in the pitching synclines that the principal ore-bodies of the district were deposited. A transverse section of the chief ore-producing field is shown by the diagram, Fig. 3, in which the later minor faulting has not been taken into account.

These inclined beds have been slightly folded, faulted in at least three different directions, repeatedly broken through by eruptives, and covered by lava-flows.

3. *Faultings.*—The recognition of extensive stratigraphic dislocation in the Lake Valley district is very recent. Clark,[9] in his description of the mine-workings, explained everything in the way of geologic structure by postulating the existence of folds. Only in two isolated instances did he even suggest

[9] *Trans.*, xxiv., 138 to 167 (1894).

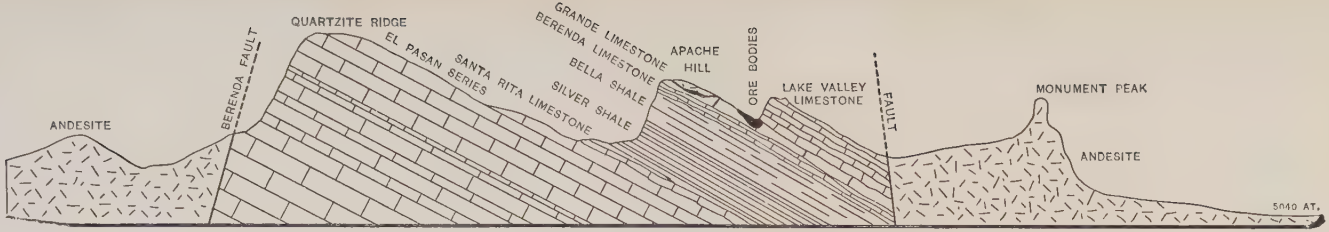


FIG. 3.—GEOLOGIC CROSS-SECTION OF LAKE VALLEY DISTRICT.

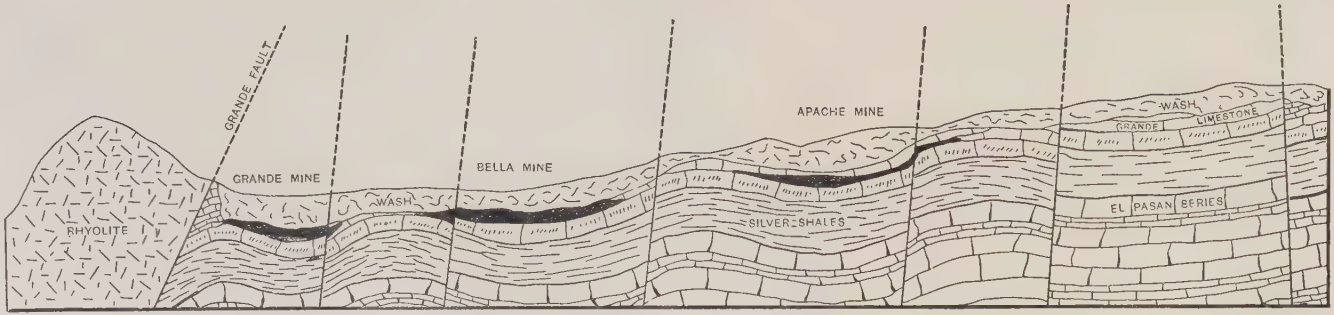


FIG. 5.—CROSS-SECTION OF GRANDE-BELLA-APACHE ORE-BODIES, SHOWING FAULTING.

the presence of faults; and these were regarded as of such small consequence as to require no explanation whatever.

Mr. Clark's account of the district contains much of value, and his descriptions of the developments are very complete, but his geologic observations are not equally clear. From his map, the real geologic structures are very hard to understand, and many representations are impossible. Frequent faulting, the evidence of which is plainly discernible on the surface, he has entirely neglected. In fact, the three most important and fundamental features, directly bearing upon the deposition of the ore-bodies and the formation of the ores themselves, have been wholly overlooked.

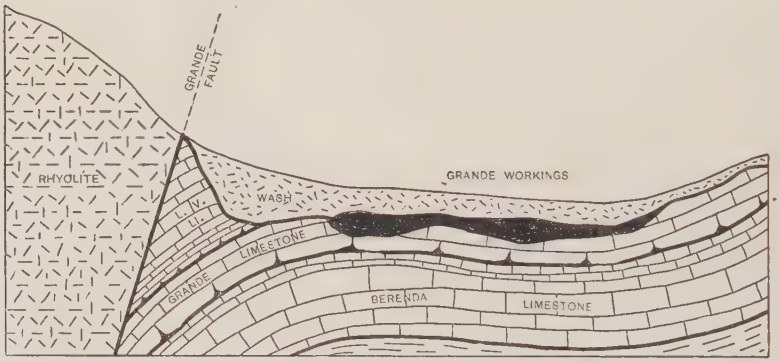


FIG. 4.—DETAILS OF THE GRANDE FAULT.

In the Lake Valley area, there have been three distinct periods of faulting, resulting in three series of faults. The first has a northwesterly and southeasterly strike, at a considerable angle to the axes of the folds already described. The second courses are nearly at right angles to the first—namely, northeast and southwest. The third series, of minor, close spaced faults, strikes due east and west.

The first-mentioned, or "Grande" fault, brings the early rhyolitic outflow down against the Paleozoic strata, cutting them off sharply. This line of dislocation passes through the Sierra Grande property and limits all exploratory work to the south. The throw of the fault has not been determined; but the indications are that it is not less than 1,000 ft. It hades about 15° south. Details of the dislocation as shown in the main workings are represented in Fig. 4. This fault appears to ante-date, by a long interval, the Berenda slip.

The second period of profound dislocation is represented by the Berenda fault, producing the Berenda valley, the Quartzite ridge and other marked features of the local topography. It appears probable that a parallel fault of minor importance lies on the west side of Apache hill, and that the long valley here is not due wholly to differential weathering. Still another parallel fault of great extent lies to the west of Monument peak, bringing the andesite flows and the Carboniferous limestones into juxtaposition. A minor fault of the same series may also exist between the Apache hill fault and the one last mentioned. The displacement of the Berenda fault is more than 800 ft.; that of the Monument peak fault at least 300 ft.

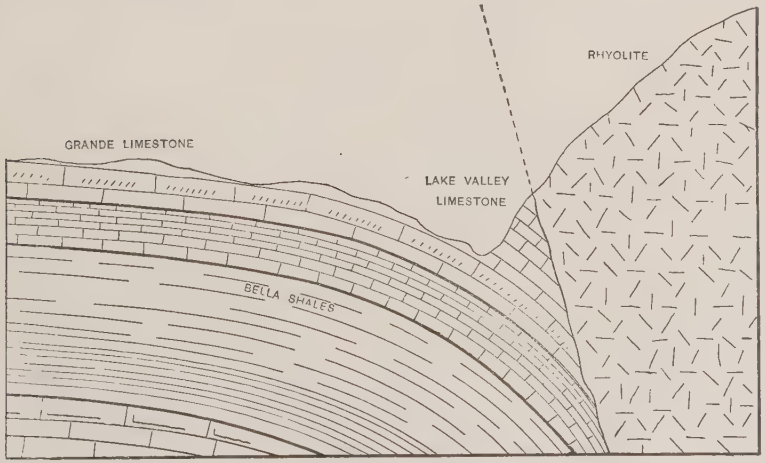


FIG. 6.—JUNCTION OF RHYOLITE AND LIMESTONE AT THE GRANDE FAULT.

In the faults of the third series the throw is nowhere great; probably in none of them does it exceed 100 ft. Nearly a score of distinct slips of this series, all of them trending east and west, and in most of them plainly traceable on the surface for a greater or less distance, have been recognized within a mile. They directly affect the ore-bodies in a remarkable way. A cross-section through the main workings is shown in Fig. 5.

4. *Relations of the Eruptives.*—The earlier, or rhyolitic, eruptive mass may have been laccolitic in character, rather than a simple surface-flow, with Lufkin mountain as a center. Through the Grande fault the northern side of the main rhyolitic mass abuts directly against the Paleozoic limestones, the

line of junction being nearly a straight line (see Fig. 1). The relations of the two rock-masses are shown in the accompanying sketch (Fig. 6).

The later, or andesitic, eruptive masses are manifestly surface-flows of lava, rather thin in places, but at Monument peak at least 400 ft. thick. They appear to have covered the entire surrounding area, except Lufkin mountain. That the limestones were completely covered, is shown by remnants of considerable size which lie upon the highest parts of Apache hill and other neighboring elevations. The relations of the Apache area of andesite to the underlying limestones are indicated in Fig. 7.

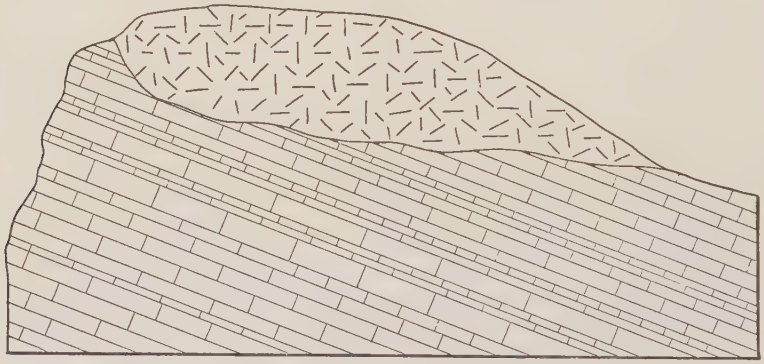


FIG. 7.—RELATIONS OF ANDESITE AND LIMESTONES ON CREST OF APACHE HILL.

VIII. GEOLOGIC RELATIONS OF THE ORE-BODIES.

1. *Governing Factors.*—The dependence of ore-deposition upon local geologic conditions is illuminated in a remarkable manner in the Lake Valley district. The ore-bodies are strictly confined to a limited geologic horizon; they are associated with a distinct lithologic zone; they are segregated in well-defined pitching troughs; they present definite relations to certain topographic features; and they are sharply delimited with reference to antecedent faulting.

These several features could be made the basis of an extended and highly suggestive discussion; but in the present connection they can be only touched upon.

2. *Position.*—All the ore-deposits are definitely situated in the local geologic column, being confined to the Grande, or Blue, limestone, which is only 25 ft. thick. On the whole, this

lime-rock is very pure, fine-grained, and easily soluble; hence, it is readily made cavernous.

In the field the Grande limestone is sharply demarcated. Immediately below it, the nodular Berenda limestone is readily distinguishable wherever encountered. Above it, the Lake Valley shaly limestone cannot be mistaken. Both of these formations have manifestly exerted an important influence upon the confinement of the ore-bearing waters to the Blue layer.

3. *Relations to Foldings.*—As already intimated, all the ore-bodies are very definitely associated with the broader corrugations of the older stratified beds in which they occur. The ore-materials have been segregated in the shallow synclines, which now pitch at an angle of about 20°. It is believed that these troughs were primarily involved in the accumulation of the ores.

It is difficult to determine just how far the localization of the ores has been due to the synclinal structure, and how far to the impounding tendencies, below ground-water level, of certain of the major faultings. The latter factor is probably the more important; but the suggestiveness of the former cannot be overlooked. The relative effectiveness of the two may be estimated by comparing the amounts and the proportional richness of the ores available at the north and south ends of the mining belt respectively. Towards the north end the main deposition of the metalliferous deposits is still clearly in the troughs, with only slight amounts on the crests of the arches. The ratio of the amount of metal from this part of the belt, as compared with that of the south end, where the impounding has been most pronounced, is about as 1 to 4.

The marked influence of shallow synclinal structure in the localization of workable ore-bodies has been noted in many other mining regions; and the recognition and determination of these features at Lake Valley should be made a directing factor in exploratory work.

4. *Relations to Lines of Dislocation.*—At first glance it would appear that faulting had disturbed and broken the main ore-bodies; and this seems to be the opinion prevalent among the miners. Closer inspection, however, shows clearly that this is really not the case. On the contrary, it is manifest that even

the latest movements of the rock-masses, represented by the minor east-and-west series of faults, were all produced before the ores were segregated in their present position. As a rule, the amount of displacement is not great enough to separate very far the ends of the ore-bearing Blue limestone, or to prevent the cavernous zone from being continuous, though it may, perhaps, be greatly reduced in thickness along the planes of slipping.

That the last period of faulting took place prior to the concentration or accumulation of the ore-bodies in the present positions is clearly demonstrated by the abrupt, straight walls limiting them along the lines of rock-movement. If the faulting had taken place subsequent to the ore-deposition, the ore-bodies themselves would be faulted; and evidence of this is nowhere apparent.

The only effect of the Grande fault at the south end of the ore-belt has been relatively to raise the hard, impervious rhyolite above the level of the cavernous Blue limestone (which is also inclined towards the fault), and thus to impound the underground waters in the zone where the ores have gathered. It is noteworthy that no ores have remained above the level of the impounded waters, as determined by the outflow over the present surface at the lowest point on the fault-line.

So far, then, as concerns any effect upon the ore-bodies after their formation, the faults may be disregarded in all mining operations within this district.

5. *Relations to Present Surface-Relief.*—The relation between ore-bodies and the local topography is so obvious that more than ordinary stress may be placed upon the existing surface-conditions. It is seldom that any dependence of the formation of ore-bodies upon the local topography can be recognized. But, to the contrary, Lake Valley presents an exceptionally interesting instance, suggesting, indeed, a possible genetic interrelationship of ore-deposits, commonly overlooked hitherto.

On the assumption that the principal mineralization of the district occurred at the time of the great rhyolitic eruptions, when the limestones were lying flat, or nearly so, there appears to have been, after the main tilting, a gradual migration of the ore-materials of the Blue limestone down the slope and a gradual enrichment, until the present location was reached.

The ore-bodies manifestly remained near the surface, but about at permanent water-level, and were covered only by the clayey limestones of the Lake Valley formation. The ridge formed by the last-mentioned terrane migrated eastward more than 2 miles to its present position. During the long period of this movement and wearing-away it served as part of a dam, as it were, which impounded the ore-bearing waters. The zone of maximum ore-deposition, therefore, advanced closely upon the constantly moving ridge.

The conditions above described remained constant until quite recently. As the great impounding ridge continued to change its position to the eastward it finally met, at its present location, the headwater erosion-lines of east-directed drainage-ways. These have cut the barrier ridge at three points, making channels through which impounded underground waters are drained eastward (instead of southward through the former overflow-point on the Grande fault-line, where the rhyolite and limestone juncture is lowest). The new drainage has not yet progressed far enough to permit the full outflow of waters down to the level of the present base of the ore-bodies; but it has been sufficient to remove the greater part of the original mass of ore. Probably nine-tenths of the former ore-mass had been already removed in this manner before man, by more rapid means, began to take away his remaining one-tenth.

Extensive exploratory work beneath the ridge of the Lake Valley formation clearly shows that ore-deposition does not extend very far beyond the margin of the clayey limestone already referred to, or the basset edges of the basal layer of that formation, as it rests on the Blue limestone beneath.

The relation of the ore-belt to the eroded edge of the Lake Valley limestone is more clearly displayed in Fig. 8.

6. *Relations to Eruptive Rocks.*—So far as observable, the ore-bodies are now in no way directly connected with the numerous eruptive masses of the region. The original contact-mineralization, near the present Berenda fault, is 2 miles away from the ore-belt. This appears to be the nearest point of direct metamorphic action.

Clark,[10] in his description of the mines, makes much of a

[10] *Trans.*, xxiv., 138 to 167 (1894).

so-called "porphyrite" overlying the ore-bodies, as the source of the ore-deposition, or as greatly influencing it. Careful examination, recently made, seems to indicate pretty conclusively that the so-called porphyrite is not an eruptive, but an accumulation of rock-waste, having lime as a large factor in the cementing material. In that case the "porphyrite" could have had no influence whatever on the formation and localization of the ores.

7. *Relations to Lithologic Nature of Country-Rock.*—That the ore-materials have been confined to a rock-stratum of distinctive character is doubtless due, partly, to the peculiar lithologic

FIG. 8.—RELATIONS OF ORE-BODY TO BASSET EDGES OF LAKE VALLEY LIMESTONE.

nature of that stratum. The selective tendency of the Blue limestone may be ascribed to a number of features. By reason of its pure calcareous character, it is liable to become cavernous; and, in connection with this condition, it is quite likely that the hydrocarbons and organic matter contained in the limestones had much to do with the reduction of the metalliferous salts in the circulating waters, and thus with the localization of the deposits.

IX. MINERALOGIC CHARACTER OF THE ORES.

The silver-ores of Lake Valley are black in color, crystalline to earthy in texture, and easily treated. They consist largely of ferric and manganic oxides, through which the other metal-

lic minerals are scattered. There is very little gangue. The iron and manganese obscure everything else contained in the deposits, and also cover the upper surface of the Blue limestone, wherever it is exposed, for a long distance beyond the margins of the workable ore-bodies.

The silver-values are chiefly in the form of cerargyrite (silver chloride), argentite, and argentiferous galena. There are also smaller quantities of other silver-ores: embolite (silver chloro-bromide), proustite (arsenious silver sulphide), and stephanite (antimonious silver sulphide). Beautiful crystals of cerargyrite occur—perfectly formed cubes about 1 mm. in size.

The fact that in the reduction-works large roasters were installed suggests that the sulphides were present in much larger proportion than casual inspection would indicate.

Other interesting mineralogical occurrences are certain vanadium compounds, and pyrolusite in beautiful crystals, botryoidal and reniform masses.

There are probably considerable amounts of carbonate ores of lead, and possibly zinc, present in the ores; but, if so, they are entirely disguised by the ferruginous minerals.

X. Origin of the Ore-Deposits.

1. *General Statement.*—The ores of the Lake Valley may be, for all practical purposes, classed as silver-chloride deposits. By far the larger proportion of the great production was of this nature. Considerable amounts of the chloro-bromide (embolite) were also mined. The formation of this class of ores in large amount is of exceptional interest, because the process finds its most favorable conditions in arid and semi-arid regions. Other climatic conditions do not permit results permanent enough to be advantageously investigated. The presence of sulphides in these deposits is suggestive. They are most abundant towards the bottoms of the ore-chambers. In the early days of mining, the carbonates of lead and zinc are said to have been abundant.

The origin of the Lake Valley ore-deposits has been the subject of wide discussion. The earliest opinion attributed the deposits directly to ascending currents of warm waters, carrying the metalliferous salts in minute quantities. A later view ascribes the ore-bodies to the leaching of an overlying "porphy-

rite," which was assumed to be silver-bearing. Other opinions have been expressed from time to time. None of these various hypotheses appear to be confirmed by the more recent examinations. In fact, all the data obtained militate strongly against these views, and indicate almost conclusively a different mode of formation.

2. *Original Source of the Ore-Materials.*—No true fissure-veins are yet known in the ore-belt, or in any part of the country for several miles around. The eruptive masses which have broken through the sedimentaries in Late Tertiary and Quaternary times have extensively mineralized the beds with which they have come in contact. Neighboring districts display this phenomenon much more clearly than Lake Valley. In some of them, the mineralization extends into the clastic rocks at least 500 or 600 ft. Some of the metalliferous contact-zones are sufficiently rich in silver sulphides and argentiferous galena (both very finely divided and disseminated) to form workable bodies.

There is also good reason for believing that, before the rhyolites and andesites broke through, and while the limestones were yet lying flat or nearly so, the Blue limestone was already a lean ore-horizon carrying lead and zinc, with perhaps a little silver. It may even be that workable deposits of galena existed thus early in the shallow synclines which were already a feature of the geologic structure.

The principal contact-zone appears to have been near the Berenda fault-line, about 2 miles west of the present ore-belt and parallel to it. All the limestone of this part of the area seems to be very much more metamorphosed and mineralized than ordinarily would be expected, and much more than are the limestones of the same age elsewhere in the neighborhood.

The great rhyolitic mass of Lufkin mountain, which cuts off the ore-belt at the south, shows no evidences of metamorphic action at its juncture with the clastics.

Remnants of thin andesitic flows are found near the summit of Apache hill, above the ore-belt, about 500 ft. vertically higher than the top of the ore-bodies. These surface-flows of acidic lava carry minerals of silver, lead and other metals in small quantity. Since they once covered probably the whole district before the valleys were eroded to their present depths,

considerable amounts of mineral may have been furnished by the decomposition of the lava sheets.

In the main, the supplies of ore-materials are believed to have come originally from the contact-veins, but to have been concentrated and enriched under unusual conditions, by means of which the final deposits were so far removed from their former position that, at first glance, no connection between the two would be suspected.

3. *Contact-Metamorphism of the Region.*—Little need be said here concerning the contact-phenomena, since these are now but remotely and indirectly associated with the existing ore-bodies. The latter, as remarked above, are now fully 2 miles from the probable main contact-zone, from which their materials were mainly derived. It is not improbable that the contact-influences extended laterally along certain stratigraphic horizons. In New Mexico it is everywhere apparent that the mineralizing effects of contact-metamorphism extend much further from the plane of contact than has been generally suspected.

As a rule, students of petrography have greatly underestimated the potency of contact-metamorphic action in connection with ore-deposits. They have been content with a contact-alteration zone of a few inches or a few feet, and have placed the greatest stress upon the effects of regional metamorphism. This circumstance is doubtless due largely to the fact that most of the more exhaustive petrographic investigations relating to this subject have chanced to be in localities in which regional metamorphism has been most in evidence, and contact-metamorphism has produced minimum effects.

In New Mexico, and the Southwest generally, and, perhaps, throughout the Mexican plateau and Great Basin regions also, the effects of regional metamorphism are practically *nil*. Contact-metamorphism only is encountered; and, being unobscured by regional metamorphic effects, it can be investigated by itself.

In the presence of so clear a display of contact-metamorphism, many of our older notions of this phenomenon are likely to undergo radical change. Instead of a simple mineralogical rearrangement of component particles, through a very limited zone bordering the contact, according to the con-

sensus of opinion heretofore, we find an extensive transference of new materials into the wall-rocks. In the sedimentary rocks, this introduction of extraneous material may reach into the walls hundreds and even thousands of feet, instead of only a few feet or even inches as a maximum. Moreover, some layers are much more affected than others; so that, for instance, in a given limestone series, the transference of even metalliferous particles may extend, in some layers, for a mile or two from the contact-plane, while in adjoining layers appreciable changes may not extend more than a few hundred feet, and in still others, not more than a few inches.

4. *Process of Ore-Concentration.*—Segregation and enrichment at the bottom of the oxidized zone probably took place in the Lake Valley deposits in the same way as in ordinary steeply-dipping veins. But this instance is peculiar, in that, after general degradation of the area had gone on for a considerable period, the limestones were disturbed and tilted at a high angle, and, as the permanent water-level was gradually lowered with the progress of general erosion, the enriched ore-bodies, through oxidation, solution, and reprecipitation as sulphides, changed their positions laterally as well as downward. In other words, they migrated, as it were, down the slope of an inclined layer of cavernous limestone, until finally they became entirely separated from the original contact-vein. A number of similar cases could be cited from neighboring districts.

The influence of geologic structure and of special features of surface-relief in bringing about this result has been already discussed.

The geologic conditions which have so long conspired to preserve the Lake Valley ore-deposits are now fast changing; and soon every trace of the original ore-belt will have been obliterated. Such inroads have been made upon the peculiar topographic feature which has served to impound the waters hitherto, that it may not be very long before the ore-zone will be drained above ground instead of underground.

The present remaining mass of the ore-deposits is but a fraction of the former aggregate, yet from this remnant millions of dollars have been realized.

5. *Lowering of Ground-Water Level.*—This district presents evidences of recent, notable and repeated changes in the

ground-water level—a phenomenon more frequent than is generally supposed, but accompanied, in this instance, with an unusually clear exhibition of the causes producing such changes.

The extent of the most recent lowering of the water-level exceeded 1,000 ft. So far as the ore-bodies are concerned, this lowering has kept in advance of the general degradation of the district. At the present time, a point has been reached in which the local erosion has just overtaken the lowering of the underground-water level, and now threatens the complete effacement of the ore-deposits.

Change in the ground-water level has not been always downward; there have been also slight upward movements. It is to this oscillation of the water-level that certain otherwise inexplicable features of the gossan-zone are to be ascribed. A general discussion of the phenomenon comes more properly in another place.

6. *Formation of Cerargyritic Ores.*—The prevalence of chloride ores in the weathered zone of the silver-deposits of New Mexico is of great interest. A fundamental influence upon both the formation and the preservation of large amounts of ores of this class appears to have been exerted by the dryness of the New Mexican climate, and by the presence of an abundance of chlorides in the circulating waters of the weathered zone of this region, as facilitating the decomposition, transference, and re-deposition of ore-materials, in both silver- and gold-veins. This region affords an exceptionally favorable opportunity for an exhaustive study of the transformations of the chloride ores generally.

The genesis of the cerargyritic ores has long been a theme of speculation. No hypothesis yet advanced can be made to cover all deposits of this class. They have, in fact, several distinct modes of origin; and the processes and conditions in one region may find no counterpart in those of another. The following hypotheses as to the formation of cerargyrite deserve mention here:

(1) The chloridization may have been due to small quantities of salt dissolved in the circulating ground-waters. This general proposition, early advanced by Bischof,[11] while it has

[11] *Lehrbuch der Chemischen und Physikalischen Geologie*, vol. iii., p. 808 (1866).

been accepted for a large number of localities, is purely theoretical and very primitive. It does not account for the presence of the chloride compounds in the circulating waters—which is, from a practical standpoint, all-important.

(2) The chloride ores have been formed by the action of weak saline solutions on the native metals. This phenomenon is frequently observed in many mines of the American arid regions, where a thin crust of horn-silver is often found covering a mass of native silver. Silver vessels from excavations of ancient cities are often coated with the chloride, as has been fully described by Schertel[12] and others. Old silver coins, unearthed, frequently show such a surface-change. In nature, however, this process can hardly have been an important factor in the production of chloride ores in large quantity.

(3) Chloride ores may have been produced by the action of sea-water on vein-outcrops, as urged by Mösta[13] for the silver chlorides of Chile. Of similar import are Brauns's[14] observations of the action of sea-water on the slag-heaps of the Laurion lead-district of Greece. Other examples might be mentioned. But cases of this kind must be at best isolated and exceptional.

(4) Such ores may be due to the direct action of waters from saline lakes. This mode of formation has been emphasized by Penrose,[15] in accounting for the production of horn-silver in the desert regions of western America; and the same idea, though differing somewhat in details, is expressed by Spurr[16] regarding some of the Nevada silver-ores. Another modification of the same suggestion is that of Ochsenius,[17] who postulates upraised lagoons, afterwards draining into the silver-deposits below. This explanation is also given by Sandberger[18] for certain Peruvian ore-deposits, particularly those of the Cerro de Huantajaya.

The existence of former Quaternary and Tertiary lakes in many parts of the Great Basin region, as a direct means of

[12] *Journal für praktische Chemie*, vol. iii., pp. 317 to 319 (1871).
[13] *Vorkommen der Chlor-, Brom-, u. Jodverbendungen des Silbers* (1870).
[14] *Chemische Mineralogie*, p. 367 (1896).
[15] *Journal of Geology*, vol. ii., No. 3, p. 314 (Apr.–May, 1894).
[16] *Geology Applied to Mining*, p. 286 (New York, 1904).
[17] *Die Bildungen der Natronsalpeters aus Mutterlaugensalsen*, p. 51 (Stuttgart, 1887).
[18] *Neues Jahrbuch für Mineralogie, Geologie und Paleontologie*, p. 174 (1874).

changing the ores of the gossan beneath into chloride form, as urged by Penrose, seems to me to be a wholly untenable assumption. Cerargyritic ores are usually found in the mountains, far above the highest level of any of these old lakes. These mountain-ores are the only ones thus far exploited. Chloride deposits, if they exist at all beneath the old lake-basins, remain to be discovered. It cannot be assumed that saline lakes formerly existed above the mountain-ores, or that the rock-masses of the mountains have all been upturned and uplifted since the lakes ceased to exist. It is true that some of the Tertiary lake-beds, 3,000 or 4,000 ft. in thickness (as, for instance, those of the Funeral range in southeastern California), have been tilted and raised to even greater heights; but cerargyritic ores do not appear to be particularly plentiful in any of these localities.

(5) The chloridizing agents may have been derived from saline materials liberated by the weathering of rocks, as incidentally suggested by Beck.[19] Spurr[20] also mentions this hypothesis, especially in connection with the deposits of Nevada. This source of suitable chlorides for the metals is much more important in humid than in arid districts. In the southwestern United States its contribution must be inappreciable.

(6) The formation of metallic chlorides may have been aided by the deposition of dust, blown from the sea-coast. This is mentioned, as a possible co-operative agency, by Beck.[21]

(7) The necessary chloridizing compounds may have been furnished by the evaporation of ground-waters in an arid climate, leaving a saline deposit near the surface of the soil. In a similar way lime-salts are extensively formed throughout dry regions, a foot or two beneath the surface, producing a deposit easily mistaken for limestone, and termed *caliche* by the Mexicans.

(8) Chlorine may have been contributed through direct volcanic activity, either by emanations of free chlorine, as in the cases of many modern volcanoes, such as Vesuvius and Cotopaxi (the latter especially noted by Wolf[22]), or by emanations

[19] *The Nature of Ore-Deposits* (Weed's translation), vol. ii., p. 375 (1905).

[20] *Geology Applied to Mining*, p. 286 (New York, 1904).

[21] *Loc. cit.*

[22] *Neues Jahrbuch für Mineralogie, Geologie und Paleontologie*, p. 164 (1878).

from dry fumaroles during the period of most energetic activity, when the temperatures are very high and anhydrous chlorides of various kinds are given off in great abundance, as described by Sainte-Claire Deville[23] and others. Such chlorides may accumulate between layers of lava, as in the cases cited by Geikie[24] of the old volcanoes of Scotland.

(9) Through the agency of wind-blown saline dust (not necessarily from the sea-coast), which is distributed in large quantities throughout the arid regions.

In the Southwest of this country the chlorides which have produced metallic ores are not thought to be directly due to volcanic activity. Notwithstanding the vast outpourings of eruptives in the region, the period of the latest of these is manifestly too remote. Although it is well known that in many volcanic regions great amounts of the more common chlorides are often interbedded with the ordinary lava sheets, as just stated, yet, in the region under consideration, general erosion has taken place to such an extent that this source cannot be looked to for an adequate supply of such chlorides.

The most likely source of sufficient supplies of chlorides to act upon the metalliferous deposits of the region is, therefore, the air. This hypothesis, the last of those enumerated above, suggests the cause which has acted most effectively of all in the Lake Valley district, and probably throughout the dry desert regions generally, though not to the total exclusion of others.

The supplies of saline materials could easily be, and no doubt mainly were, wind-borne. The dusts of New Mexico always contain appreciable quantities of saline particles. Dust-storms are violent and frequent. While they last, the atmosphere is so filled with fine soil that it is impossible to see more than a short distance. Ephemeral pond-waters, in drying up, often leave a white coating on the ground that is popularly termed "alkali." In reality, the white precipitate is composed very largely of common salt.

The amount of saline material in the dusts of New Mexico is a constant factor. The quantity of salty matter furnished

[23] *Annales de Chimie et de Physique*, Third Series, vol. lii., p. 19 (1858).

[24] *Proceedings of the Royal Society of Edinburgh*, vol. ix., pp. 367 to 371 (1875–8).

by this means very greatly exceeds, for a given area, that supplied by the decomposition of the rocks. When the rock-layers are highly inclined, as they are in most of the basin ranges, the saline matter in the dust deposited on the surface is easily dissolved by the first rain, and largely carried below, to mingle with the underground waters.

The introduction of measurable amounts of saline material into the ground-water manifestly supplies one of the most favorable conditions for the formation of haloid compounds of the metals. Another favorable condition is a geologic structure which promotes the impounding of meteoric waters underground; and a third is such a peculiar surface-contour as aids in gathering saline materials, retaining them in the surface-waters, and preventing them from passing off before they have opportunity to be included in the waters underground. In the Lake Valley and neighboring districts, all of these conditions combine in high degree to the production of abnormally large amounts of saline matter in the waters circulating through the gossan.

Other haloid compounds of the metals are also found in the Lake Valley deposits. The chloro-bromide of silver, embolite, occurs in sufficient abundance to form an important percentage of the ores mined. Were it not for the large amounts of manganese and iron oxides which make the entire ore-body appear to consist merely of ores of those metals, it is probable that other haloid minerals would be recognized in greater or less abundance. Several such minerals are known to occur in neighboring districts, where their presence is not obscured by these dark-colored materials. The bromide and the iodide of silver (bromyrite and iodyrite) are instances; the chlorides of lead and of copper may also be present. A careful inquiry into the mineralogy of these ore-bodies would prove highly instructive.

No native silver has been reported to occur at Lake Valley; yet this does not preclude its occurrence in considerable quantity. In the neighboring districts, under similar conditions (except that the associated manganese and iron oxides are not so abundant), large amounts of native silver are found. It may be suggested that the cerargyrite has been derived from such native silver, as has been found to be the case elsewhere (and is, indeed, probable in an instance only a few miles away); but

the shallowness of the Lake Valley deposits, their position near permanent water-level, with only a few feet of wash above, and the presence of abundant sulphides in the deeper parts of the deposit, preclude this supposition.

On the other hand, the presence of sulphides, the thinness of the deposit (which may be regarded as a "blanket-vein"), the character of the local relief and geologic structure, both of which are especially fitted for the purposes of collecting and retaining meteoric waters, indicate that the cerargyrite was formed directly from sulphates, produced by the oxidation of the sulphides (argentite, stephanite, and perhaps also silver-bearing galena).

In a steeply-dipping vein, the migration of ore-material downward through the oxidized zone is well known. It is also often the case with cerargyrite, the richest ore-bodies of which are frequently found near ground-water level, or immediately above the sulphides. In other instances, such as those described by Mösta,[25] in Chile, there appear to be several distinct zones in which cerargyrite occurs. In general, the downward journey of the chloride ores in such veins is not difficult to follow; but the Lake Valley deposits in this downward journey have been, after a while, chiefly diverted down a steep and open stratigraphic plane, instead of the line of the original vertical contact.

XI. Recapitulation.

The most suggestive features brought out in the foregoing discussion may be briefly summed up. In the Lake Valley district it appears that:

a. The original source of the ores was in extensive contact-mineralization.

b. The ore-deposits have definite relations with the geologic structure of the district.

c. The ore-deposits are also fundamentally dependent upon peculiarities of the local surface-relief.

d. The ore-deposition has shown a marked preference for certain rock-layers.

e. The ore-deposits have now no direct association with any of the numerous eruptive masses of the region.

[25] *Vorkommen des Chlor-, Brom- u. Jodverbindungen des Silbers* (1870).

f. The ore-deposits have not been faulted to any appreciable extent.

g. There has been marked oscillation in the ground-water level.

h. The usual explanations of the origin of haloid ores are of very limited application, and probably do not obtain at all in the southwestern United States.

i. The main source of the chlorides producing the cerargyritic ores has been the saline dusts of the arid regions.

j. In the arid regions eolian agencies predominate among the geologic factors, while in humid regions aqueous agencies are most effective.

Diamonds in Arkansas.

BY GEORGE F. KUNZ, NEW YORK, N. Y., AND HENRY S. WASHINGTON, LOCUST, N. J.

(New York Meeting, February, 1908.)

THE recently discovered occurrence of diamonds near Murfreesboro, Pike county, Ark., was brought to our attention by Mr. Samuel W. Reyburn (Trustee for Messrs. C. S. Stifft, A. D. Cohn, August Zinsser, Jr., and himself, owners of the property), through whose courtesy we investigated the locality of the deposit. Prospecting with the diamond-drill is now in active progress; but mining operations on a large scale have not yet been ordered.

Geology.—The geology as well as the petrography of this interesting locality has been well described by Messrs. J. C. Branner and R. N. Brackett,[1] but they did not class it as diamondiferous, although Branner suggested that it might be. Briefly summarized, the igneous rock in which the diamonds are found is a vitreous peridotite, forming a stock or volcanic neck, which has broken up through the Carboniferous and Cretaceous quartzites and sandstones. After an extensive period of erosion, during which an unknown portion of the neck and pre-

[1] *American Journal of Science*, Third Series, vol. xxxviii., No. 223, pp. 50 to 59 (July, 1889); and *Annual Report of the Geological Survey of Arkansas for* 1890, vol. ii., pp. 377 to 391 (1891).

sumably a previously existent volcanic cone have been removed, the surface was covered with thin beds of Post-Tertiary conglomerate. The volcanic intrusion was accompanied by the formation of several small dikes of a rock much like that of the main body. One of these dikes cuts across the stock, while another cuts the Cretaceous sandstone, but is overlain by the conglomerate, thus giving a datum for the period of intrusion. So far as known, there was little, if any, metamorphism of the country-rock by the igneous magna, which probably followed an approximately vertical course, so that a more or less vertical extension downward of the igneous body to indefinite depths may be expected. This result should hold good, at least, for the upper and most accessible portions, though some departure from strict verticality may be expected at greater depths.

As already observed, the igneous rock is a peridotite which, in fresh hand-specimens, is tough, hard, distinctly porphyritic, and very dark greenish- or brownish-black. Microscopic study shows it to be composed of numerous crystals of olivine and some patches of biotite, imbedded in a ground-mass of very small crystals of augite, perofskite, and magnetite, with an abundant yellowish to colorless glass base. In all the specimens examined by Mr. Brackett or by us the olivines are more or less completely serpentinized, and the glass is apt to show an aggregate polarization due to decomposition. The rock is evidently an igneous intrusive, which probably welled up in comparative quiet, and solidified not far from the surface. It is therefore in no sense a volcanic breccia, due to explosive eruptions, as are most of the South African occurrences. Chemically and mineralogically, however, it much resembles the South African rock, although there are certain points of difference—notably the absence of inclusions.

Peridotites are generally prone to alteration by weathering. In this instance the freshest rock is dense, hard and tough, and does not crumble markedly on exposure, as is evidenced by the fact that the highest points in this igneous area are exposures of fairly-fresh peridotite. The first state of pronounced alteration is the disintegration of the firm rock into a mass of hard angular fragments, varying in size from that of a bean to that of a human fist, which apparently do not readily disintegrate on exposure to the weather. The second stage of

alteration, due to further weathering, yields a compact mass, the so-called "green ground," showing various shades of light olive-green, and often bluish in tint when moist, but becoming yellowish on drying. The third state of alteration, found nearer the surface, furnishes, from still further oxidation of the ferrous iron, the so-called "yellow ground," which resembles the "green ground" in physical characters, but is, in color, distinctly brownish-yellow, with little or no trace of green. The green ground and the yellow ground are soft and friable, crumble readily between the fingers, and show soft, but sharply defined, serpentinous pseudomorphs of the original olivine crystals, with well-preserved outlines. This fact, supplemented by the general appearance of the texture, shows clearly that the peridotite has been decomposed in place, and that there has been little or no transportation of the material.

Both the green and the yellow grounds, if dry, crush under a gentle pressure to a fine powder, containing small gritty particles of the less-decomposed minerals, which can be readily sifted out. If wet, the rocks disintegrate rapidly, especially with mechanical agitation, to a fine, somewhat sticky mud, which can be easily washed or otherwise concentrated.

The fresh, compact peridotite crops out at the surface, forms several small hills along the NW. border of the deposit, and is also visible at other points; and the first fragmentary alteration-product shows itself at a few spots; but the green and the yellow grounds are found over by far the greater part of the igneous area, either on the surface or, more frequently, immediately beneath a thin layer of black, sticky, "gumbo" soil. The maximum and average depths of this mass of decomposed peridotite have not yet been exactly ascertained; but borings show it to be, in places, 40 ft. thick. This fact, together with other considerations, leads us to estimate the average thickness to be not less than 20 ft. Below this is found either the fragmentary, or a more or less compact, igneous rock. One drill-hole has penetrated the peridotite to a depth of 205 ft., another to 186 ft., and a third to 80 ft.,—all remaining in igneous rock to the end, as was to be expected, in view of the geologic structure.

The surface exposure of the igneous area forms a rough ellipse, about 2,400 ft. in major and 1,800 ft. in minor diameter.

The area known to be underlain by peridotite is estimated at about 40 acres, though further prospecting of the neighboring alluvium-covered bottom-land to the south may possibly add to this amount. The limits in other directions are more clearly marked.

General Conditions.—A variable supply of water, usually abundant, is furnished by the Little Missouri river, which flows a short distance to the southwest of the igneous area. This stream, though somewhat low at certain seasons, never runs dry, and may be safely counted on to provide a sufficient supply of water for all mining purposes. For certain installations, however, its rapid and sometimes serious rises must be taken into consideration. The owners of the igneous area possess, also, a large tract of land, along both sides of this stream, with the incident water-rights.

A large portion of the land under control is well wooded; and extensive forests, chiefly of pine, with some oak, promise a good supply of cheap timber for some years to come.

Coal may be readily purchased at a reasonable cost from the bituminous fields of Arkansas, Indian Territory, or Texas.

Although the region is not thickly settled, and the nearest towns are small, the experience of the lumber-companies indicates that an ample supply of labor (chiefly white) will be available. Indeed, the lumber-camps themselves may be an immediate source of supply. In this connection an obvious, and possibly serious, difficulty may be mentioned—namely, the prevention of the loss of diamonds through theft by the laborers. With the class of labor employed at the South African mines, a system of detention in compounds, thorough physical examination for hidden diamonds, and other methods for the prevention of theft or the recovery of stolen stones, can be carried out; but in the United States it might be impossible to employ, at least in a thorough-going manner, safeguards of this character. Up to the present time, a small force of picked men having been employed in the preliminary operations, and about 140 diamonds having been found, there is little or no ground for the belief that any serious loss of this kind has occurred. But work on a large scale, involving the employment of a large number of laborers of less trustworthy character, and increased difficulties of adequate supervision,

will augment this risk, the prevention of which will be a serious problem.

Transportation facilities for coal, machinery and other supplies are furnished by two short branches from the Iron Mountain Railroad. One (a private lumber road) leaves the main line at Prescott, and extends 26 miles to Nathan, about 6 miles from Murfreesboro, while the other runs from Gurdon on the main line about 30 miles to Pike City, distant about 10 miles from Murfreesboro. Only very rough roads now connect these terminals with the diamond-bearing locality; but the improvement of these roads, or the extension from Murfreesboro of another railroad, may be safely counted upon in the near future.

Factors to be Determined.—Up to the present time about 140 diamonds have been discovered within the igneous area, while none have certainly been found outside of it, even in the immediate vicinity. All the stones have been found on the surface, except two, which were in the concentrates derived from washing large amounts of the green ground, and one, which was imbedded in the green ground itself about 15 ft. beneath the surface. Our careful examination of this last specimen, confirmed by Dr. R. W. Raymond, leaves no doubt that the diamond is actually in place in the rock and was not inserted in the specimen. Consequently it constitutes a definite proof that the peridotite is the source of the diamonds, and that all the stones so far discovered have been derived from it. It would be well, however, to have this single piece of evidence corroborated by similar specimens. With regard to the quantitative relations of the diamonds to the inclosing rock, about 200 carats have been found on or immediately beneath the surface, where presumably there has been considerable concentration of the stones. From the nature of the deposit, the average yield per ton can only be ascertained by actual washing or other extraction from the rock on an extensive scale, commensurate with that of purposed commercial operations.

Additional factors of economic importance, for which more extensive data are necessary, are the average size, color and quality of the stones, since these factors determine their value. From the 200 carats at present available for examination, it appears that the Arkansas locality compares very favorably

with most, if not all, of those in South Africa. Although no stones larger than 6.5 carats have yet been found, the average size is fairly good. There is a large proportion of white stones, for the most part of a high grade in color, brilliancy, and freedom from flaws. Indeed, many are as fine as have ever been found. Some of the yellow ones, also, are of exceptional quality and color.

A more exact determination of the limits and extent of the workable area, based on a large survey and on extensive prospecting with the drill, is now being made.

The method of extraction of the diamonds is of vital interest and importance. The green and the yellow grounds offer no difficulty, and are amenable to the methods used in South Africa. Indeed, in Arkansas, there is no need for prolonged exposure to the weather, since the freshly extracted material disintegrates and can be washed with ease. The amount of this easily worked material "in sight" is very large; yet, it is not of indefinite extent downward, as is the "blue ground" of Kimberley; and, consequently, its extraction will form but a transient phase of future exploitation.

The economical extraction of the diamonds from the compact, and relatively fresh and hard, peridotite, underlying the "green ground" and forming the vast bulk of the mass, will involve study, and probably experiment. But, apparently, there will be no greater difficulties than have been successfully overcome in South Africa. In view of the hard, tough, and fresh character of the peridotite which composes the highest points of the area along the NW. border, it might be thought that the material underlying the green ground would be of the same character and equally refractory; but the diamond-drill shows that, at least for considerable depths, a large proportion of the underlying peridotite is far more decomposed than that which crops out at the border—is, indeed, so far altered that much of the material comes up as sludge, and no continuous cores longer than 14 in. have been obtained. Many of these cores were so soft as to be readily scratched with a knife. Probably this more compact material will disintegrate on exposure to the weather, like the South African "blue ground." If this proves to be the case, a large proportion of the mass will not be difficult to work.

At some portions of the mass, however, as at the NW. border, and presumably in depth beneath the rest of the area, fresher and much more refractory material will be encountered, the treatment of which will present practically the same problem as that of the hard portions of the African rock which do not disintegrate on exposure. While a certain amount of crushing, in order to extract the diamonds, is apparently unavoidable, this should be reduced to a minimum, on account of a high loss from breakage of the stones themselves. Several methods of treatment suggest themselves, which are at present under advisement; but the practical details, as well as the economic features, remain to be worked out and cannot be discussed here. The non-magnetic character of the diamond and its tendency to adhere to grease are obvious features which can undoubtedly be used at certain stages of the extraction for all classes of material in the Arkansas deposit, as in South Africa.

A Word of Warning.—In view of the great local excitement over the discovery of diamonds, which has extended over part of the State, and in view of the danger of the repetition here of the disastrous history of many mining camps which have undergone an unwarranted "boom," and the consequent rush and loss of time and money by many innocent individuals, it should be distinctly understood by the public that the occurrence of diamonds near Murfreesboro is an isolated one, and that it does not resemble a mineral vein or lode in any respect. Consequently, there is not the least justification for any such claims as will undoubtedly be made by ignorant or unscrupulous parties, that "a continuation of the vein" has been struck. There can be no continuation of a vein when there is no vein.

Should other similar igneous areas, which may possibly be diamond-bearing, be discovered elsewhere, any claims put forward for them should be received with the greatest caution. Fortunately, the characteristics of the peridotite (in which, by analogy, diamonds may be most reasonably expected to occur) are so easily recognizable by a petrographer, the localities will be presumably so isolated, and the outlines and extensions of the mass so well defined, that the report of a geologist or petrographer can surely prevent an unsuspecting or ignorant person

from loss by investment in a property said to be a continuation of, or a connection with, the present deposit.

Peridotites are not uncommon; but very few are diamond-bearing. Indeed, the great majority of these rocks found all over the world show no trace of diamonds. Even in South Africa, many peridotite pipes, resembling valuable ones, carry no diamonds, while in any diamond-bearing pipe some portions are found to be richer in diamonds than others.

As shown by J. F. Kemp,[2] many basic dikes have been found in Arkansas; but most of these differ petrographically from the Murfreesboro peridotite, and there is no reason to think that any of these, or any of the several syenitic areas of the State, are connected with diamond-bearing rocks. As has been noted above, two dikes of peridotite occur in connection with the Murfreesboro igneous area. Great stress is laid locally on these dikes, or "leads," as they are called, but without warrant, since there is no reason to think they contain diamonds, and in any case they are too small to be of economic value. From analogy with other igneous intrusions, it is probable that more dikes will be discovered in the neighborhood, radiating out from the main stock; and in other localities the presence of dikes of similar rock, which could only be identified by petrographical means, would be an indication of the possible presence of a larger body of peridotite in the vicinity. If diamonds are present, they are to be looked for in the rock-mass itself, or in its products of weathering, and not only along the contacts, because they are integral portions of the igneous mass, and their presence is not due to the circulation of hot water and solutions along the contact between an igneous mass and the country-rock.

[2] J. F. Williams, *Annual Report of the Geological Survey of Arkansas for* 1890, vol. ii., pp. 392 to 406 (1891).

The Central Power-Station of the De Beers Consolidated Mines, Ltd., Kimberley, South Africa.

BY PERCY A. ROBBINS, NEW YORK, N. Y.

(New York Meeting, February, 1908.)

I. Introduction.

The central power-station of the De Beers Consolidated Mines, Ltd., was designed and built under my supervision about five years ago. Since no detailed description of this plant has ever appeared, it may possess sufficient interest to warrant its publication, even at this late date. Although some additions may have been made to the plant, the following account, prepared from my notes, and accurate up to May, 1905 (at which time I severed my connection with the De Beers Company), will, I think, be found substantially correct.

Certain local conditions governing the design of this plant should be stated first.

II. General Considerations.

The climate is hot and dry. The region around Kimberley is arid and water is very scarce. The supply obtained from the mines contains so much material in solution that it is unfit for use in boilers or condensers (although tests recently made with a water-softening plant, built by Hans Reisert, of Germany, would indicate that water thus treated is suitable for boilers). And the amount of water required for concentrating about 30,000 tons of ore per day is so large that the question of adequate water-supply for this power-station had to be carefully considered.

The rainy season extends from December to April, and the rainfall averages about 15 in. per year. Droughts are frequent; and in some years there is scarcely any rain at all.

The water-supply for the town of Kimberley is brought from the Vaal river, 16 miles distant, and pumped into town by a private water company. The charge made for water, which is quite suitable for boilers and condensers, is 16c. per 100 gallons.

The cost of fuel, per ton of 2,000 lb., landed in Kimberley, was, in 1905, as follows: Colonial coal having 60 per cent. of the calorific value of Welsh or Pocahontas, 42s. 6d. ($10.20). Natal coal having about 90 per cent. of the value of Welsh, 63s. ($15.12). Welsh and Pocahontas, about £7 10s. ($36.45).

Recently, however, a cheaper coal has become available in Kimberley, owing to a new railway-line which was constructed after I had left South Africa.

Practically all supplies were imported from Europe or America. The sea-freight was about 45s. ($11) per ton; and the railway-charges from the coast to Kimberley about 8s. ($2) per 100 pounds.

Burnt bricks, made locally, cost £4 ($19.50) per 1,000 landed at site of the power-station.

Portland cement was imported, and cost from 26s. to 30s. ($6.24 to $7.20) per cask of 400 lb. landed at the power-station.

There was no local building-timber supply. All timber for the De Beers Company was imported from the Pacific coast of America, and cost, delivered at Kimberley, $1 per cu. ft. Fire-bricks were imported from Scotland. All glass, structural steel, piping, mechanical and electrical appliances were imported from America or Europe. Mechanics, masons, etc., received from $4.50 to $5 per day; and shop-work cost from two to five times as much as in England.

The foregoing facts are given merely to show the necessity for economy in design, construction and operation. Fortunately, a supply of native unskilled labor was available at a rate of wages of from 80c. to $1 per day.

In consequence of the high cost of work done locally, the distance from the nearest base of supply (7,000 miles), and the length of time required for obtaining the materials for the plant, it was necessary to design every detail at the outset in order that nothing should be lacking, and that the plant, as a whole, should fit together when erected.

III. Situation of the Plant.

A site, about central in regard to the distribution of power, was chosen at the side of a small pond known as Blankenbergs vlei, a shallow depression into which drains the storm-water from a catchment-area of about 6 sq. miles.

Data obtained from some seven years of observation showed that the rate of evaporation of water exposed to the atmosphere and the sun's rays averaged about 0.25 in. per day throughout the year, and during certain months amounted to $\frac{3}{8}$ in. per day. The greatest depth of water known in the pond was about 6 ft., and, starting with this depth, and considering evaporation to take place down to a depth of 2 ft. 6 in., the average surface exposed would be about 1,250,000 sq. ft., and the total evaporation during the six months of the dry season would be 27,344,000 imperial gallons.

By building a dam across the middle of the pond and storing the water at a depth of 11 ft. on one side of the wall, the average surface exposed was reduced from 1,250,000 to 800,000 sq. ft., which reduces the evaporation to 17,500,000 gal., a saving of 10,000,000 gal., which, with water at 16c. per 100 gal., represented a saving of $16,000 per annum.

The dam cost about $3,500, and was built from mine-tailings, the side slopes being pitched with waste rock.

The spot chosen for the power-station site is on a limestone cap about 4 ft. thick, overlying a rather soft shale. It was decided to build on top of the limestone as far as possible, so as to save cost of foundations.

Figs. 1, 2 and 3 show the general arrangement of the power-station and apparatus.

IV. Buildings.

The main features in designing the buildings were cheapness and utility. In view of the location of the plant, architectural effect was not a consideration.

A brick engine-room was necessary to protect the more delicate apparatus from the effects of the severe dust-storms frequent in this district. For the boiler-house, a steel frame, covered with galvanized iron, was adopted; this type being cheaper than brick. The footing-courses for the engine-house were of basalt, taken from a nearby quarry, and stepped into the surface

of the limestone cap. There is a foundation-course of hammer-dressed rock up to an average height of about 6 ft. above ground-level, and above this, the building is of local brick, laid

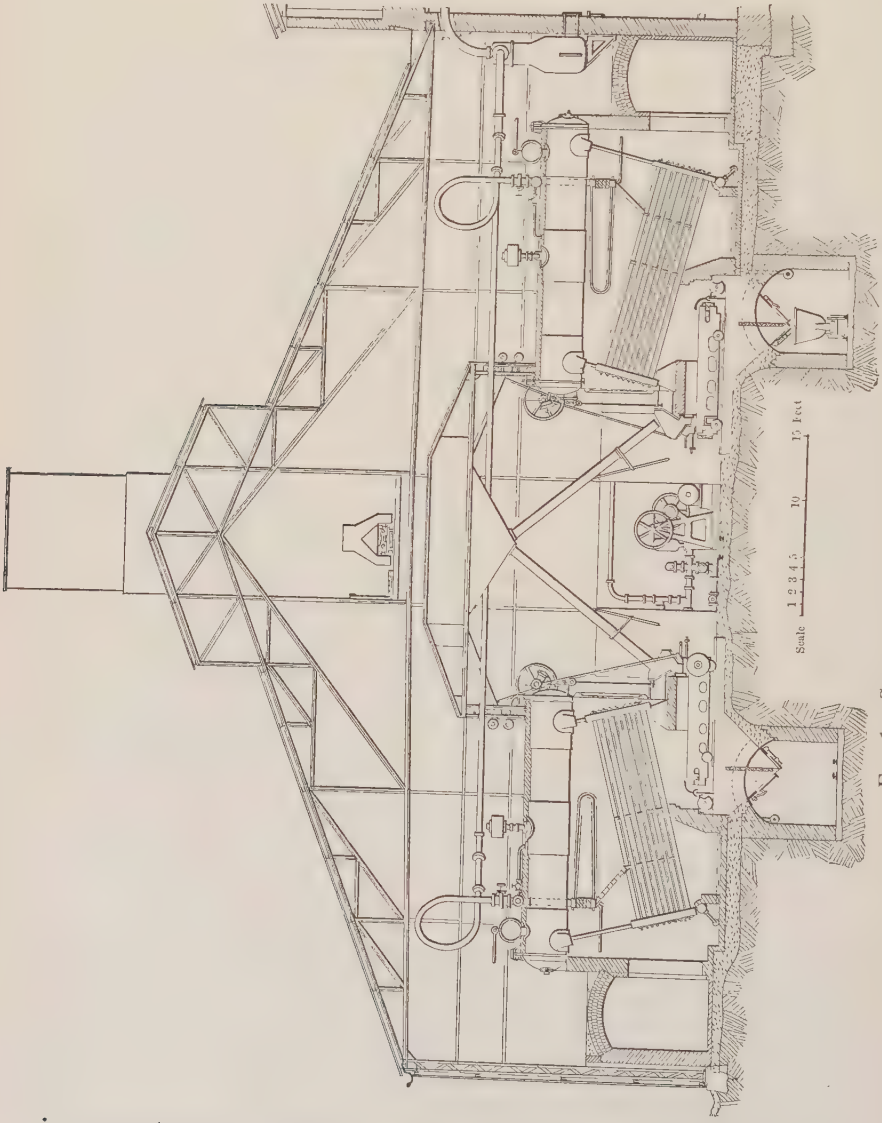


FIG. 1.—SECTION THROUGH BOILER-HOUSE.

in cement-mortar, mixed six to one. The bricks used throughout South Africa lay up roughly 3 by 4.5 by 9 in. The division-wall between the engine- and boiler-houses was made 18 in. thick, buttressed at intervals to 27 in. thick. A series of

five arches was sprung along the turbine-room side of this wall to support the rail for the electric traveling-crane which spans the engine-room. It was originally intended to carry this rail

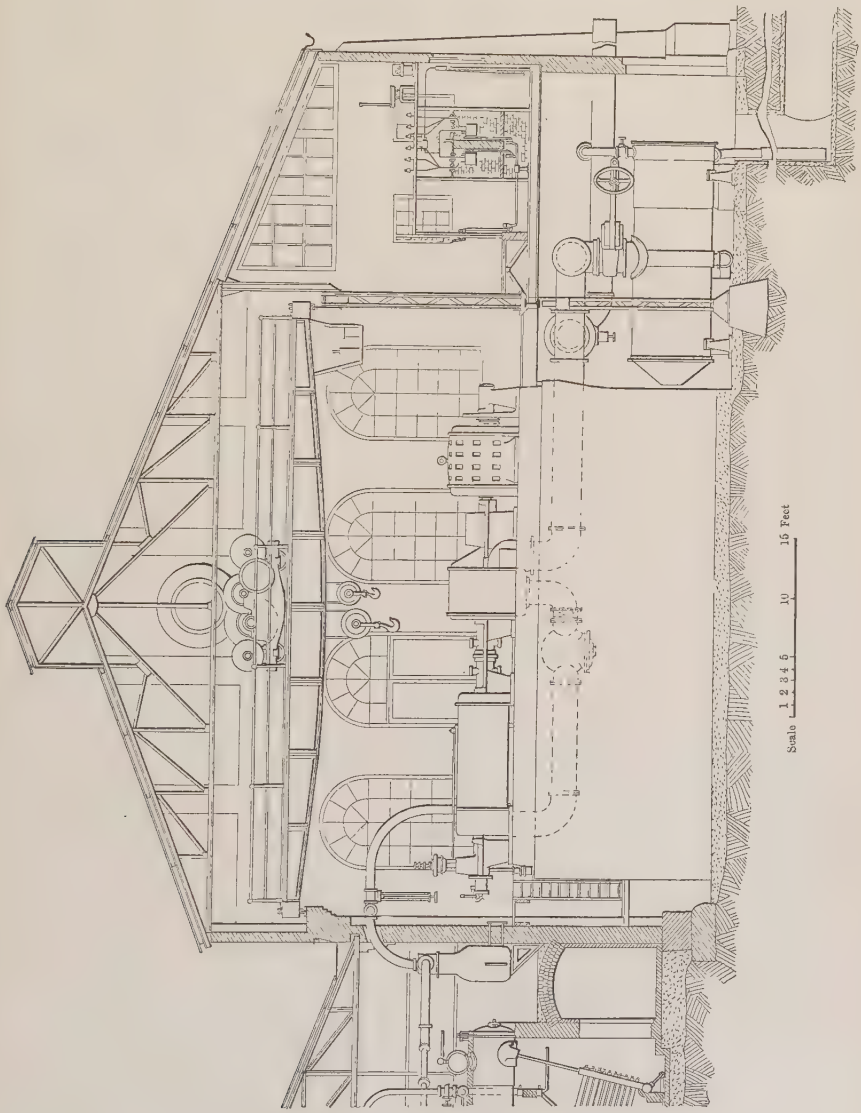


FIG. 2.—SECTION THROUGH TURBINE- AND CONDENSER-ROOMS.

on a steel girder resting on buttresses, but, owing to a misunderstanding, this girder was not supplied by the contractors for iron-work, and hence it became necessary to form the arches above referred to. The floor of the switchboard room is sup-

ported at one end by the wall of the building. At the inner end it is carried on steel girders supported by columns resting on concrete blocks at the level of the condenser-room floor. These supporting-columns extend upward to carry the 24-in. steel crane-girder; 8-in. channels are carried up from the height of the crane-girder to support one end of the engine-room roof-

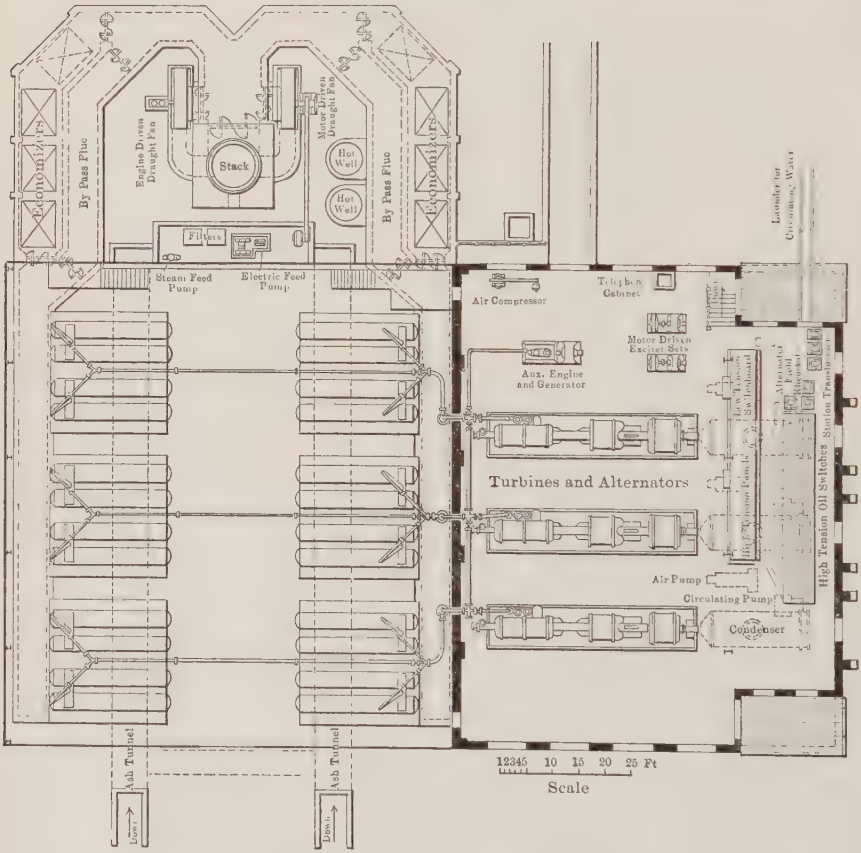


FIG. 3.—PLAN OF POWER-PLANT.

trusses. These 8-in. channels also support the inner end of the steel rafters of the lean-to roof which covers the switchboard room. The engine-room roof-trusses are mild steel, designed to carry the dead weight of the roof plus the live load due to a wind velocity of 100 miles per hour. The roof is covered with No. 24 galvanized corrugated iron, and is ceiled inside with 4 in. by 0.75 in. tongued-and-grooved boards. This wood

ceiling was necessary to avoid the condensation of moisture at night and on cold days. If galvanized iron only is used, the moisture collects in drops and drips down upon the apparatus underneath. A monitor carried by the roof is provided with windows which may be opened or closed from the engine-room floor.

The windows, door-frames, etc., in the brick engine-house were made according to drawings sent to New York, and the building was erected with openings left to receive the fixtures when they arrived. The windows are covered with wire netting, in order to protect the glass from being broken by the terrific hail-storms which occasionally pass over the country.

The boiler-house consists of a steel frame covered with No. 24 galvanized corrugated iron. The roof-trusses are supported at one end by templates carried in the division-wall between the turbine- and boiler-houses, the outer end being carried on latticed columns. The roof of the boiler-house is not ceiled. The floor in the boiler-house was laid in concrete.

The floor of the engine-room is of wood, although it was my intention to replace this floor with reinforced cinder-concrete carried by steel girders. The floor of the switchboard room is of concrete, laid over sheet-iron forms sprung between the supporting floor-girders. The condenser-room floor is like that of the boiler-house.

V. Coal-Handling Plant.

The coal-cars are pushed into a siding and the coal unloaded by hand over the side of the car into heaps. Small 20-cu. ft. side-tipping mine-cars are loaded by hand from these heaps and then pushed around to the coal-crushers. Plans had been developed for mechanically unloading the railway-cars, stacking the coal and afterwards removing from stacks to crushers, but this work had not been carried out at the time I left Kimberley. The crushers consist of two pairs of pony-rolls made by the Dixon Manufacturing Co. These rolls have hardened steel, diamond-pointed teeth which break the coal up into a size suitable for burning—namely, not greater than 1-in. cubes. One pair of rolls is set slightly further apart than the other. This is found necessary in crushing different grades of coal, as certain classes of harder coal break too fine when passed

through close-set rolls. The rolls discharge on to a 20-in. Robins belt-conveyor, which rises at an angle of 21° until it reaches the height necessary to discharge into the hoppers over the boilers. So long as there is a sufficient quantity of coal on the belt the coal rides quietly and does not roll back, but when the belt is nearly empty, small round pieces have a tendency to roll about and eventually fall off. This same difficulty, experienced also in another plant I designed, was overcome by driving the belt from the bottom end, and making it horizontal for about 10 ft. at the point where the coal is delivered to it. The trouble in the present instance, however, was not sufficient to warrant the expense of making this change. The coal is discharged from the belt into hoppers by means of an automatic tipper supplied by the makers of the belt.

Sufficient bin-capacity is provided to hold an 8-hr. supply of coal. The bins are designed for stresses of 14,000 lb. tension, 12,000 lb. compression, and 10,000 lb. shear on rivets. The coal is drawn from the bins to the furnace-hoppers through square chutes, shown in Fig. 5.

The chutes are spread at the lower ends into two branches. At the junction of these branches with the main chute is fitted a gate-valve for cutting off the flow of coal. The branch chutes are made detachable in order to permit the boiler-fronts to be swung open.

VI. Boilers.

Boilers of the Babcock & Wilcox type had been used extensively by the De Beers Co. in previous work, and were thus thoroughly understood by the employees of the company. The straight tubes facilitate repairs, cleaning and inspection.

There are 12 boilers, each of 3,580 sq. ft. of heating-surface, set in six batteries.

Each boiler comprises 16 sections or slabs, containing 10 best, wrought-iron, lap-welded tubes, 4 in. in diameter, and 18 ft. long, connected at the ends by continuous wrought-iron staggered headers, or "up takes" and "down takes;" the tubes being fastened therein by being expanded. Each "header" is provided with hand-holes placed opposite the end of each tube, of sufficient size to permit the cleaning, removal

and renewal of a tube and having a cap fastened with wrought-iron bolt and clamp and cap-nut. The several sections are connected at each end to two steam- and water-drums, and at one end with a mud-drum, by means of wrought-iron lap-welded tubes, 4 in. in diameter and of suitable length, the joints being made with an expander.

The steam- and water-drums are 42 in. in diameter and 23 ft. 7 in. long, made of mild steel plates $\frac{7}{16}$ in. thick, the longitudinal seams being double riveted; at one end is a manhole, also a nozzle for a safety-valve and a second one for taking off steam, 5 in. in diameter, with 12 in. flange, faced and drilled.

The two drums for each boiler are joined at the rear end with a steel-riveted cross-drum 20 in. in diameter, fixed to the main drums by expanded connections.

The mud-drums are of special cast metal, 10 in. in diameter and 116 in. long, with hand-holes and a nozzle for blow-off pipe, fitted with a Hopkinson 2.5-in valve.

The boilers are suspended from wrought-iron girders, resting on wrought-iron columns, with cast-iron bases properly secured so that the weight is sustained entirely independent of the brick-work. This arrangement permits free expansion or contraction and removal or replacement of the brick-work, if required, without disturbing the boilers or connections.

Each boiler is provided with two "dead-weight" safety-valves set to blow at 147 lb., one steam-gauge, dial 12.5 in. in diameter, two sets of asbestos-packed Reflex water-gauges, two 2.25-in. feed check-valves, one 2-in. feed stop-valve, one 2.5 in. Hopkinson blow-off valve, one 0.75-in. cleaning-valve, and one 5-in. main stop-valve.

The fronts of the boilers are of ornamental pattern, containing large doors for access to the ends of the tubes, and the exposed brick-work is white enameled.

The boilers were designed for a working-pressure of 145 lb. per sq. in.

The tube sections and mud-drum were tested and made tight under a hydraulic pressure of 300 lb. per sq. in., and the steam- and water-drums under a hydraulic pressure of 225 lb. per sq. inch.

Each boiler is fitted with a Babcock & Wilcox patent superheater designed to superheat the steam 75° F. It is an open

question in my mind whether it would not have been advantageous to install separately-fired superheaters so as to permit independent control over the temperature of the steam. However, the superheaters have been operating for more than two years, and no trouble has been experienced with them.

Each furnace is fitted with two chain-grate stokers supplied by the makers of the boilers, and different qualities of coal have been burned with equal facility, the stokers giving perfect satisfaction during the period covering the running of the plant, a fact all the more striking when it is considered that no men experienced in the operation of mechanical stokers were available at the time the plant was put into operation.

These mechanical stokers possess the inherent fault that they permit too much cold air to get into the combustion-chamber. If, however, as in the present instance, the boiler-capacity is ample, a low draft may be used, and with careful attention to the rate of feed and thickness of fire, a condition of efficient working may be maintained.

The ashes are delivered over the inner end of the stokers and fall into a hopper over the ash-tunnel. The fine coal which falls through between the stoker-bars is pushed by hand, from time to time, into a separate compartment of the ash-hopper. The ashes are drawn out through one door and the fine coal is drawn out through another. Originally it was intended to provide a mechanical ash-handling plant, but it was decided that a mule, on the end of a rope attached to the ash-cars, makes the most economical and inexpensive ash-conveyor for a plant of this size. The cars containing the ashes are drawn up an incline to the surface by the mule, trammed out to the ash-dump and there tipped. The fine coal is drawn up an incline to the surface and tipped into the coal-handling plant, by which it is elevated to the hoppers above the boilers.

The gases from the boiler pass into flues built on ground-level, and thence to one end of the station where the economizers are placed. The flues are of ordinary burned brick, lined with 4.5-in. of similar brick bonded at intervals into the flue-walls. As plan, Fig. 3, shows, the boiler-house is divided into halves, each containing six boilers, a flue common to all

six boilers, a set of economizers, and a draft-fan which discharges into the base of a stack, which latter is common to the entire boiler plant. The economizers are by-passed by flues extending at one side. Where necessary, the thrust of the flue-arches is taken up by tie-bolts bolted through washer-plates on the outside of the flue-walls. Inside of the flues, where exposed to the action of the gases, these tie-bolts are protected by wrought-iron pipes. Access to the flues is had through cast-iron doors built into the brick-work. The flues are carried around into the base of the stack, as shown in Fig. 3, in order to bring the draft-fans inside of the area adjacent to the end of the boiler-house.

There are two draft-fans, supplied by B. F. Sturtevant Co., each inclosed in a three-quarter housing, one driven by an engine direct-connected to the fan-shaft, while the other is belted to a 30-h.p. induction motor. These fans are located on opposite sides of the stack-base, the electrically driven one generally being used.

The stack-base, of burnt brick, has no fire-brick lining. The stack consists of an outer shell 10 ft. in diameter, built up of steel plates with lap joints and lined throughout its length with 9 in. of ordinary burned brick, a 2.25-in. air-space being left between the brick lining and the plates. Although not necessary, since its base was sufficiently broad to resist overturning from any ordinary wind pressure, the stack was stayed with four wire-ropes anchored in concrete blocks.

The general arrangement of economizers, fans and stack is shown in Fig. 3.

As already noted, there are three sources of water-supply—namely, hard water from the mines, river-water from the Vaal river and storm-water conserved in dams.

At the time of designing the plant it was not considered advisable to use the mine-water for boiler-purposes. The Vaal river supply is led up to the power-station through a 4-in. main. The supply from the vlei is pumped into the filters by a small three-throw pump located on the condenser-room floor. The water is elevated to a filter-tank placed at the base of the stack.

The water-supply piping, shown in Fig. 4, was designed so as to bring, as far as possible, all of the valves controlling the different supplies together, in order to facilitate changing from

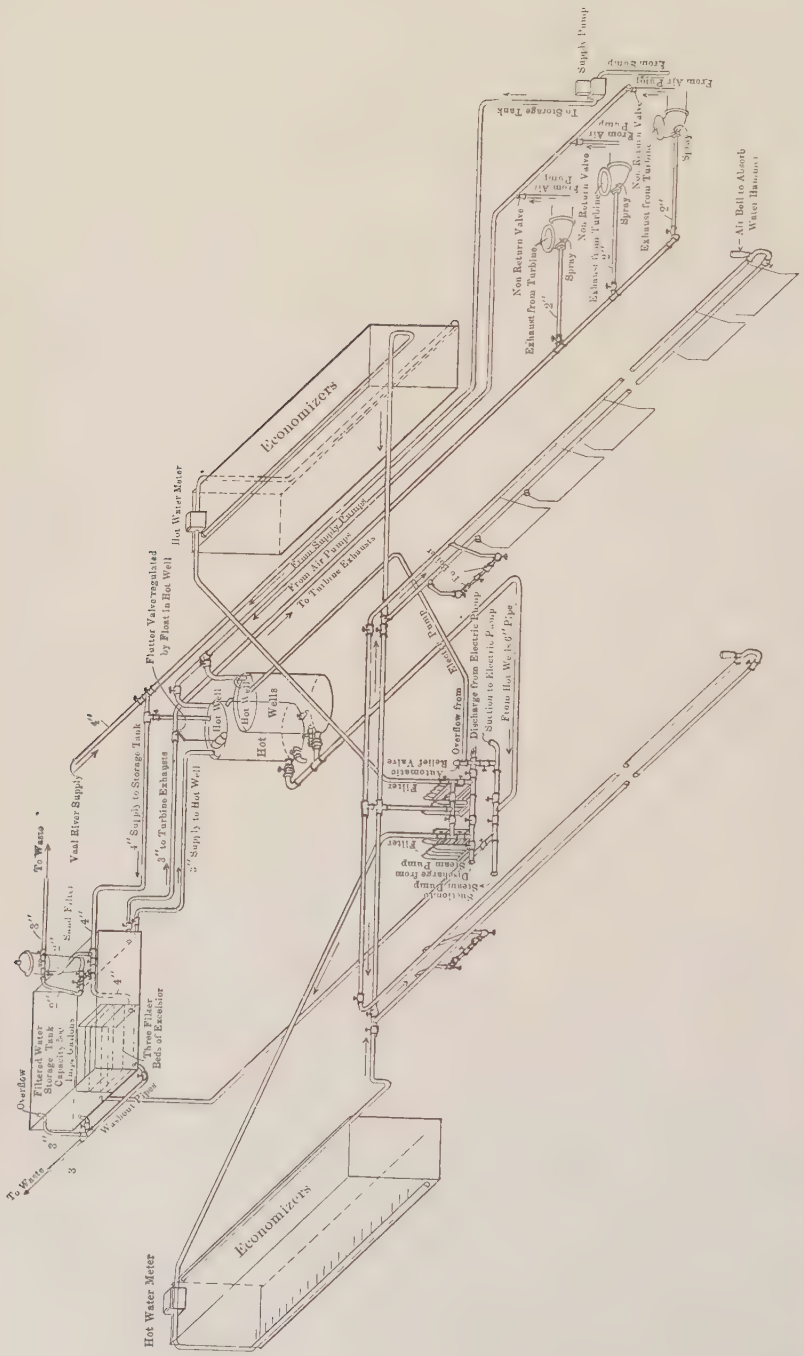


FIG. 4.—SCHEME OF WATER-PIPING.

one to another. All water which enters the station is thoroughly filtered before reaching the boilers. It first passes through a crushed-quartz sand-filter, thence to the bottom of a tank containing three coir mattresses arranged one above the other, and upwards through these mattresses, finally overflowing into the main storage-tank. The piping is so arranged that any part of the filtering-apparatus may be by-passed during cleaning or repairing. There is also a connection with the hot-well by means of which water may be led directly into the hot-well without going through the filters. There are two hot-wells arranged as shown, made of boiler-plate riveted in cylindrical form, each being 6 ft. 1.25 in. in internal diameter by 8 ft. in depth, and insulated on the outside with 9 in. of brick-work. These wells are arranged in duplicate in order to permit continuous running while cleaning. The hot-wells stand on the level of the boiler-room floor, so as to insure that the suction of the feed-pumps will be submerged. It was found necessary to cover the tops of the hot-wells with iron gratings, in order to prevent employees from falling into them. Several fatal accidents had already occurred from this cause in the works of the De Beers Company.

There are two feed-pumps, one electrically and the other steam-driven. The former is a vertical, three-throw, with outside-packed plungers 5.5 in. in diameter by 8-in. stroke, and runs at a speed of 45 rev. per min., and is geared direct to a 15-h.p. induction motor, the pump and motor being mounted upon a common bed-plate.

The delivery of the pump is fitted with an air-bell and an automatic relief-valve, the latter having its overflow connected with the hot-well in order to return to the hot-well any excess of water delivered by the pump.

The steam-pump is a Weir tandem-compound, with high pressure steam-cylinder 10 in. in diameter, low pressure steam-cylinder 18 in. in diameter and pump-barrel 10 in., all with a common stroke of 25 inches.

The electrical pump is run about 10 hr. per day, being supplemented by the steam-pump during overloads. During periods of small loads on the station the steam-pump is used.

The suctions of the pumps are below the water-level in the hot-wells, thus insuring that they shall be always flooded.

The pipe-connections between the pumps and the feed-ranges are shown in Fig. 4, the connections being so arranged that either pump can deliver its water through filters and economizers, or direct to feed-ranges above the boilers. The filters between the pumps and the boilers consist of a number of cylindrical chambers, each containing a perforated copper tube covered with a stocking of Turkish toweling. The water passing through this toweling leaves behind any impurities which may have remained after the preliminary filtering. From the feed-pumps the water passes through the filters to hot-water meters and thence to the economizers, which are in two sets, each of 320 tubes, one for each side of the boiler-house. The economizers are of standard Green construction, with vertical cast-iron tubes forced into cast-iron headers. From the economizers the water passes into the feed-ranges over the boilers. There is a double set of feed-pipes so as to avoid shut-down in event of repairs being necessary to one range. An air-bell is located at the extreme end of each range to take up any water-hammer or other sudden fluctuations in pressure. The air-bells are provided with air-discharge cocks.

The tops of the economizers are covered with asbestos mats about 2 in. thick. The scraper-gears are driven by 2-h.p. induction motors placed over the economizers, the scrapers being run at intervals during the day.

Water is delivered from the economizers at a temperature of about 220° F.

The feed piping is 4-in. heavy lap-welded pipe with heavy cast-iron flanges, and between the economizers and boilers it is insulated with a covering of flake mica, 2.25 in. thick, so as to prevent radiation of heat.

The arrangement of steam-piping, given in Fig. 1, shows that the steam from each boiler stop-valve passes through a full loop bend before reaching the straight-through connections which lead to the separators. This arrangement was followed, as I wished to avoid any thrust on the turbines, due to expansion of the piping. It will be readily understood that any expansion in the loops will tend to open them out, and cause a slight tension on the piping to the turbines, thereby relieving the turbines of any deleterious thrust. In practice it is found that the turbines, when heated, expand about $\frac{5}{16}$ in.

in length, and as this expansion is in the direction of the boiler-house any stiff arrangement of piping would be almost sure to distort the turbine-cylinders and cause trouble.

No attempt was made to provide against vibration in the piping until after a trial run had been made, so as to enable observations to be made which would insure the proper placing of the anchor-bolts and stays.

The turbines admit steam in pulsations at the rate of about 161 per minute.

Owing to the loop bends in the piping above the boilers the whole piping-system may be regarded as a pendulum. The natural period of vibration of the loops themselves would be approximately 1 sec. It would therefore be expected that when the pulsations of steam admitted to the turbines came into synchronism with the natural vibrations of the piping, harmonics would be set up in the piping which would cause maximum vibrations to take place. This was found to be true, the periods of maximum vibration being followed by periods of quiescence.

The vibration due to these harmonics was not in itself of any great magnitude; in fact, were it not for other causes it would hardly have been necessary to anchor the piping. Separators, drained by steam-traps, were installed between the turbines and their boilers, and occasionally a trap will hang up and allow the water-level in its separator to rise to a point where the period of vibration of the inclosed body of steam is such as to cause the separator to act as a resonator to the vibrations of the piping-system. Under this condition, the vibration of the piping becomes very marked.

All troubles due to vibration were eliminated by careful staying and anchoring, without interfering with expansion, and there has scarcely been a leaky joint in the steam-piping system during the two years of operation of the plant.

At full load, condensing, each turbine requires about 21,000 lb. of steam per hr., which corresponds to a velocity of slightly less than 4,000 ft. per min. in the 7-in. piping. In the 6-in. pipes, connecting opposite pairs of boilers, the flow is about 3,500 ft. per minute.

All steam-piping is covered with 2.5 in. of corrugated flakes

of mica held together in wire-netting forms, bent to fit the pipes, and covered on the outside with heavy canvas.

The steam-piping is of extra heavy lap-welded mild steel, and all bends were rolled hot from best quality of extra heavy lap-welded steel-pipe. All piping was set up and measurements checked before shipping from England, the various parts being marked with numbers indented in the metal, and all flanges prick-punched so as to facilitate identification at Kimberley. Due allowance was made for jointing with corrugated copper gaskets, which lie entirely within the bolt-circles. All joints are male and female.

The pipes and fittings were guaranteed against a hydraulic test-pressure of 250 lb. per sq. inch.

The steam from each battery of boilers passes through a Stratton separator made of mild steel plate and containing a volume of 30 cu. ft. Separators are located in the boiler-house over the flue, where the air is always warm, thus insuring a minimum of radiation, and loss of heat is further provided against by a covering of mica-plaster 2 in. thick. Each separator is fitted with two Hopkinson water-gauges, one placed in the turbine-room, its connections being carried through the wall, thus enabling the engineer in charge of the turbine-room to keep an eye on the water in the separators.

Steam-traps drain the separators into the hot-wells.

VII. Steam-Turbines.

At the time when the power-station at Kimberley was designed, the use of steam-turbines was very limited, and the only large machine which had been built for driving an electrical generator was the one supplied by Messrs. C. A. Parsons & Co. for the Electrical Supply Works of the city of Elberfeld, having a rating of 1,000 kw. at full load.

The tests on this machine, together with the data available on smaller machines, determined me in my choice of turbines for the prime-movers in the station.

Briefly, the advantages in favor of turbines for central power-station work are:

1. The cost, installed, is less than that of first-class reciprocating-engines of the same full-load efficiency.

2. At light loads, and hence for constant running 24 hr. per

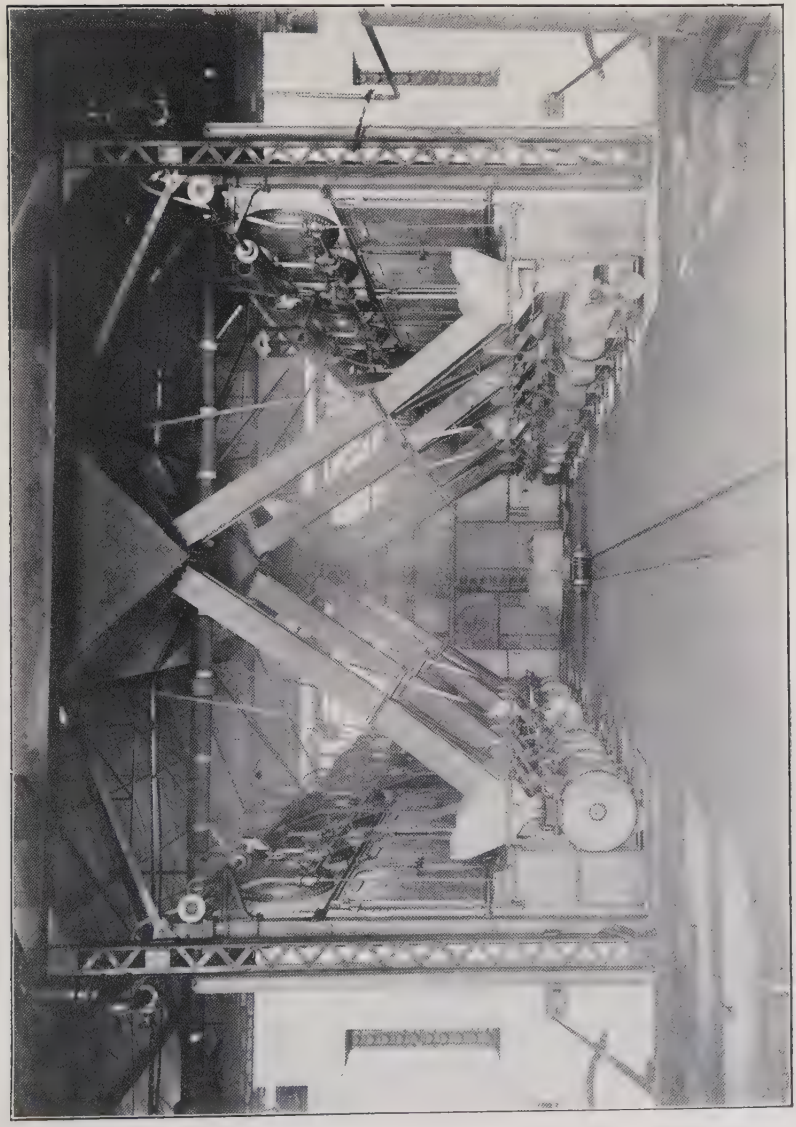


FIG. 5.—VIEW IN BOILER-ROOM.

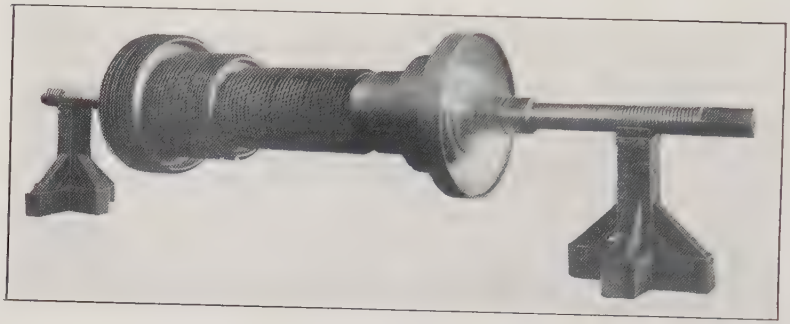


FIG. 6.—HIGH-PRESSURE TURBINE SPINDLE.

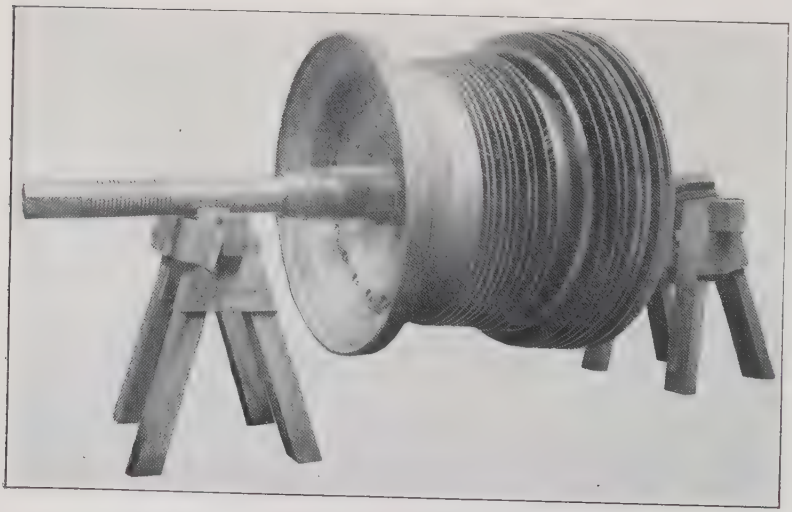


FIG. 7.—LOW-PRESSURE TURBINE SPINDLE.

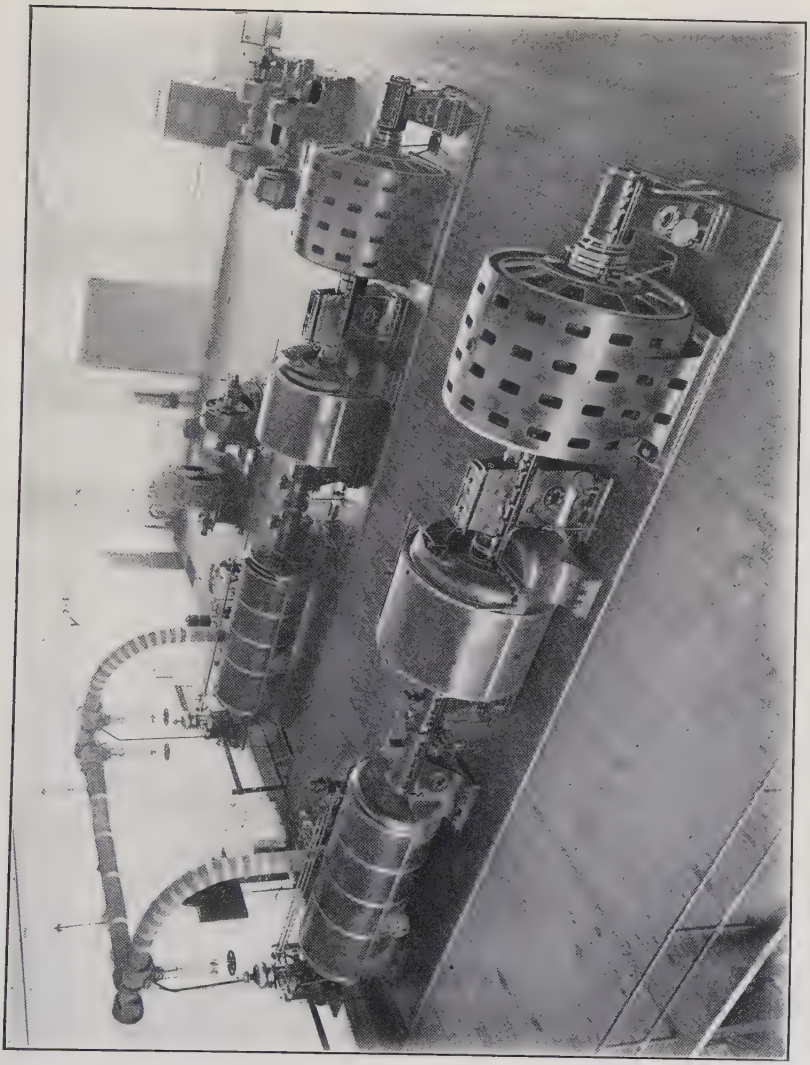


FIG. 8.—VIEW OF TURBINE-ROOM.

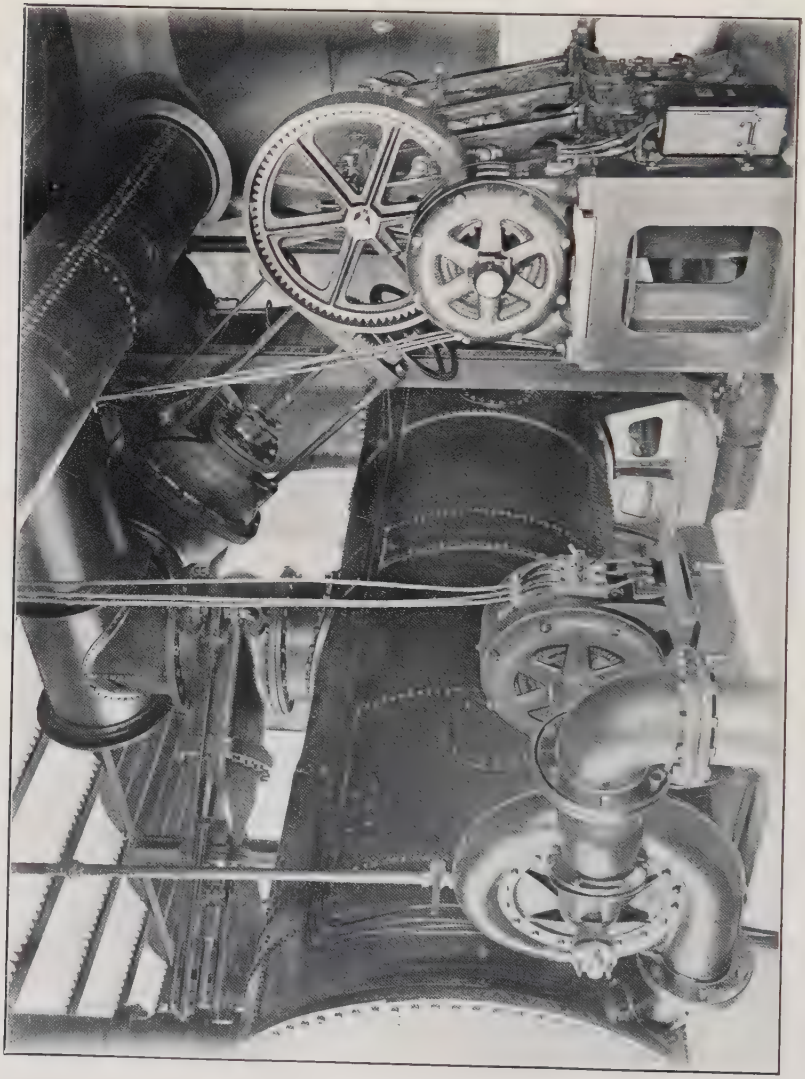


FIG. 9.—ONE CONDENSING-SET.

day, the turbine consumes less steam per kw. output than does a reciprocating-engine.

3. The cost of maintenance on turbines is considerably less than that of reciprocating-engines.

4. There is no cylinder-lubrication, and hence the water returned from the condensers to the boilers may be kept practically free from oil; the only oil which finds its way into the steam in this class of turbine being that used to lubricate the wearing-parts of the admission-valve and its relay—a very insignificant quantity.

The principal reasons, however, in favor of turbines are electrical, especially the constancy of speed, due to the high angular velocities of the moving parts. I have stood by a turbine carrying 25 per cent. overload and have failed to detect any change in the hum emitted when the whole load was suddenly thrown off, and, after an interval, thrown on again.

Careful tests show that the variation in speed between no load and full load is only 2 per cent., and when full load is suddenly thrown on or off, the turbine settles down to a constant speed in less than 2 sec. of time.

A peculiar advantage possessed by turbine-driven alternators is the fact that in a station carrying heavy inductive loads, the spare machine makes an excellent power-factor regulator when run as a synchronous motor.

In this connection I submit a few extracts from a letter from Mr. R. S. Mansel, Electrical Engineer of the De Beers Explosives Works, near Cape Town, of which I designed the power-plant, containing three 400-kw. Parsons turbines and 3-phase generators.

The extracts from the letter are:

"Up to the present our large motor-load has had a very low power-factor, rarely exceeding 70 per cent.; for this reason it has been impossible to get more than 300 kw. out of any one of our machines, owing to the excessive drop in voltage . . . . By synchronizing the spare machine, shutting off the steam in its turbine and giving its generator a heavy field-charge, we have succeeded in raising the power factor to 85 per cent., and now have no trouble in getting the full rated overload from the generators supplying power to the works."

The impracticability of using an engine-driven generator in this manner will be readily understood, on account of both the mechanical and the electrical difficulties. During my stay in

Kimberley, the opportunity of trying the experiment on the 1,000-kw. generators did not present itself, since the load on the station grew so rapidly that there was never a spare machine.

I have taken space to explain this advantage, not so much on account of its importance, but because I have never seen any published reference to this matter. It will be understood that the generator-iron must have a low flux-density (a peculiarity of turbine-type machines) in order to accomplish the desired result.

The turbines and generators at Kimberley were supplied by the Westinghouse Co. and are of the two-cylinder type. Steam is expanded through a series of blades contained in what is known as the high-pressure cylinder, whence it passes into a low-pressure cylinder. The high- and low-pressure spindles are shown in Figs. 6 and 7 respectively, while Fig. 8 gives an interior view of the turbine-room.

There are three turbine-units, each consisting of a Westinghouse-Parsons steam-turbine, direct-connected to and mounted upon the same shaft with one 3-phase alternating-current Westinghouse generator. Two of the units have a capacity of 1,000 kw. useful energy output, working under an inductive load having $\cos \theta = 0.85$. The third set has a capacity of 1,500 kilowatts.

Steam is delivered to the turbine at 145 lb. pressure and 75° F. superheat.

The current from the generators is delivered at 5,000 volts pressure.

The turbines are of multiple-expansion, parallel-flow type, each machine consisting of two turbines, arranged in tandem compound. The steam distribution is such that when working under full load the high- and the low-pressure turbines develop approximately equal power. The speed of all three machines is 1,500 rev. per minute.

The maximum length of turbine, including generator, is 42.9 ft.; maximum width, 7.8 ft.; maximum height over largest diameter, 6 ft.; height to top of governor, 8 ft. The total shipping-weight is 130,000 lb., the weight of the heaviest piece being 45,000 lb. For carrying overloads, a by-pass is provided for passing high-pressure steam into the low-pressure turbine, by means of which an overload of 50 per cent. may be carried.

The cylinders are made of close-grain cast-iron, designed so

that at no time shall the tensile stress exceed 2,000 lb. per sq. inch.

The shafts are of high-quality, mild steel, having mounted thereon cast-steel rings for carrying the turbine-blades. The maximum stress in the shaft, due to bending and twisting movements, does not exceed a fiber-stress of 8,000 lb. per sq. inch.

The blades are of bronze, calked into dove-tail grooves in the cylinders and in the steel rings upon the shaft. The longitudinal distance between the stationary blades and the revolving blades is about $\frac{5}{16}$ in. The clearance between the ends of moving blades and cylinders, and between the ends of stationary blades and rotating parts, varies from 0.01 to 0.04 inch.

The bearings are of gun-metal and consist of two concentric sleeves mounted upon the shafts, the shaft being free to revolve within the inner sleeve, and the latter being free to revolve within the outer sleeve. Thrust-bearings are provided, fitted with adjustment-screws, which permits the shaft to be moved longitudinally.

A sensitive governor is provided with each turbine, which holds the speed practically constant under all loads; the action of this governor may be assisted by hand during the synchronizing of the generators. A safety governor also is provided for each turbine, which automatically shuts off steam when the speed rises above a certain predetermined point.

A throttle-valve is also provided with each turbine.

The cylinders are lagged with loose asbestos contained in wire sacks, the whole being inclosed in cases of planished sheet-steel.

Some trouble has been experienced in lubricating the turbine-bearings, which could have been avoided if the requirements had been better understood at the outset. In order to secure efficient lubrication it is only necessary to provide means for cooling the oil after it has been through the bearings, which can be easily accomplished if there is a supply of cheap cold water available. At Kimberley, however, it was necessary to save the cooling-water, and for this purpose a small cooling-tower was erected, through which the circulating-water is dropped after extracting the heat from the oil. The oil flows through the bearings to a small pump, driven from the turbine-shaft by gearing, which forces the oil through coils placed

in the turbine-base, the cooling-water circulating around the coils.

Since the erection of an efficient cooling-tower for the water, the trouble with the bearings has disappeared, and after two years of almost constant service, the turbine-shafts and bearing-sleeves show scarely any wear, the original tool- and oil-stone marks still being in evidence.

It will be well, however, for prospective users of turbines to give the question of oil-cooling careful study.

Before shipment from Pittsburg, the two 1,000-kw. sets were thoroughly tested for steam-consumption by the late Dr. R. H. Thurston and Mr. H. H. Norris, of Cornell University, the results being as follows:

Load on generator (kw.), . .	269.7	495.5	752	1013	1247
Steam pressure (lb.), . .	143	142.5	142.5	138	140.9
Vacuum (in.), . . .	28.24	28.30	28.18	27.96	27.55
Superheat (°F), . . .	77.6	84	75.3	59	75.2
Steam per electrical h.p. per hr.,	19.27	15.58	14.73	14.22	13.67
Barometer (in.), . . .	29.2	29.25	29.3	29.3	29.2

These data make a good showing concerning the steam-consumption per e.h.p. delivered, and when it is considered that these turbines were among the first of any size built by the Westinghouse Co., the performance is all the more creditable.

The condensers for the plant were made locally, by utilizing the shells of some old fire-tube boilers. Calculations for cooling-area were based upon the assumption that cooling-water would have an initial temperature of 80° F., and an allowance was made of 4.25 sq. ft. of condenser cooling-surface per kw. of capacity of turbine. Subsequent results tend to show that a slightly greater area would have been advantageous, due to the fact that the condenser-tubes become quickly coated with a layer of slime, which detracts from their efficiency.

The condensers are counter-flow, *i. e.*, the steam enters at the top and the condensed water discharges at the bottom, while the circulating-water enters at the bottom and discharges at the top. Fig. 9 shows one of the condensing-sets and the arrangement of circulating- and air-pumps.

The in-take tunnel for the condensing-water is provided with a double screen made of perforated plates for keeping out grass

and frogs, and the suctions to circulating-pumps also are provided with screens.

The circulating-pumps are 9 in. centrifugal, each being driven by a 30-h.p. motor.

Air-pumps are of the Edwards type, each having three buckets 14 in. in diameter by 12-in. stroke, the crank-shaft making 150 rev. per min. Barrels, buckets, valve-seats, etc., are of brass. Bucket-rods are rolled bronze. Crank-shaft is mild steel, machined all over, and carried by four cast-iron "A" frames.

Each air-pump is fitted with a small water-pump, to take the water from the main pump and force it up to the hot-well.

The circulating-water from the condensers is elevated to a height of 14 ft. above the condenser-room floor, and discharged from the end of the building into an open-air launder 1,200 ft. long, made of rough sawn boards with feathered joints, having a fall of 10 in. in 100 ft. It discharges into the storage-pond at the corner farthest from the intake to condensers, a distance of 1,000 ft. By this means the water in the launder flows 1,200 ft. at a high rate of flow, turning over and over and constantly coming in contact with the atmosphere, and afterwards flows slowly across the pond back to the condensers. Cooling is thus effected largely by radiation. A cooling-tower depending upon evaporation for cooling might give better results, but the loss of water would be excessive, and water is scarce in Kimberley.

In order to utilize as much of the heat of the exhaust steam as possible, all make-up water for the boilers is injected into the exhaust-pipes from turbines, as follows:

A float is suspended in the hot-well by means of a rod attached to the crank of a flutter-valve. This valve is in a pipe-line which connects at one end with the bottom of the filtered-water storage-tank, and has branches to each exhaust-pipe immediately under the turbines. As the water-level in the hot-well drops, the flutter-valve opens and allows the cold make-up water to be injected as a spray into the exhaust steam from the turbines—an effect which has the dual advantage of improving the vacuum and at the same time raising the temperature of the make-up water. An increase in vacuum has been noted due to the use of the spray.

The exhaust-piping, shown in Fig. 2, has an atmospheric exhaust-valve placed between the main exhaust from each turbine and an atmospheric pipe, so as to prevent any extraordinary rise of pressure in the low-pressure turbine in the event of the failure of the condensing-plant.

All exhaust-piping is riveted wrought-iron, the 30-in. piping made of 0.25-in. plate, and the 24-in. pipe of $\frac{3}{16}$-in. plate. Pipes were made in halves with longitudinal divisions, the longitudinal seams being left unriveted so as to permit the pipes to be nested for shipment, the butt-straps being riveted to one of the half sections. Flanges are of cast-iron, riveted to the pipes.

All bends, tees, etc., are riveted up out of bent plates, in order to lighten the fittings as much as possible, and lessen the transportation-cost to Kimberley, which was $51 per ton.

All valves are double-seated gate-valves, fitted with renewable seats and with by-passes, the operating-spindles being geared to the valve-stems.

VIII. Electrical Equipment.

Power is delivered from the generators at 5,000 volts pressure, 3-phase, 50 cyles per sec., the generators having slot-wound stationary armatures with star-connected windings.

The revolving fields are 4-pole and the potential of the exciting current is from 80 to 100 volts. Under test, the 1,000-kw. generators developed the following characteristics:

Drop in pressure from no load to full load at 85 per cent. power-factor was 12.4 per cent.

The efficiency was, with quarter load, 82.5; half load, 90.3; three-quarters load, 93.05; and full load, 94.4 per cent.

The principal loss, as would be expected in generators of this type, is in the iron of the armature-core, this loss constituting 82.75 per cent. of the total loss at full load; or, expressed in terms of the full load, core-loss equals 4.6 per cent.

Current for field-excitation is delivered from two motor-generator sets, each consisting of one 50-h.p. induction motor, 3-phase, 220 volts, mounted on the same bed-plate with and direct-connected to a 37.5-kw. Westinghouse generator, which delivers direct current at a pressure of from 80 to

FIG. 10.—HIGH-TENSION WIRING-DIAGRAM.

100 volts. Current from the generators is led to the switchboard by triplex lead-covered cables carried under the floor.

The 5,000-volt switchboard is of the standard construction of the General Electric Co.; the wiring is shown in Fig. 10. The current for the various instruments is converted to a safe pressure by transformers placed along the back of the wall and accessible for renewing fuses from the insulated platform which extends over the wiring-connections.

Switches, immersed in oil-tanks contained in separate brick compartments, are operated through wood- and iron-rod connections, by levers on the front of the marble board.

Two sets of bus-bars are provided, each generator- and each feeder-circuit being provided with two switches, one for each set of bus-bars, which gives all the flexibility necessary in a power-station of this size. Connected with each of the bus-bars is a ground-detector, voltmeter and power-factor indicator.

Looking at the front of the board, the three generator-panels occupy the left end.

Each panel contains 1 ammeter, 1 power-factor indicator and 1 voltmeter, synchronizing-plug, synchronizing-lamp, field-switch, field-rheostat and two switch-levers, each of which operates a 3-pole oil-break switch.

Each switch is provided with a time-limit overload-relay, which automatically releases the switch and allows it to open if the current through the switch rises above a certain predetermined value.

An incandescent lamp, immediately over the handle of the switch-lever, lights up when the switch automatically opens, thus attracting the attention of the switchboard attendant.

The station-panel, mounted next to the generator-panels, contains a double set of instruments, one for each set of bus-bars, each set comprising three ammeters, one on each phase, for indicating the current passing, and one Thomson integrating wattmeter for recording the power delivered to each set of bus-bars. A ground-detector for each set of bus-bars is also mounted upon this panel.

Adjoining the station-panel are the five feeder-panels, each containing three ammeters, a recording wattmeter and two switches, whereby each circuit may be connected to either one or both sets of bus-bars.

One feeder-panel is reserved for station use and controls a circuit supplying current to a bank of three 100-kw. oil-cooled Westinghouse transformers, which transform the supply from

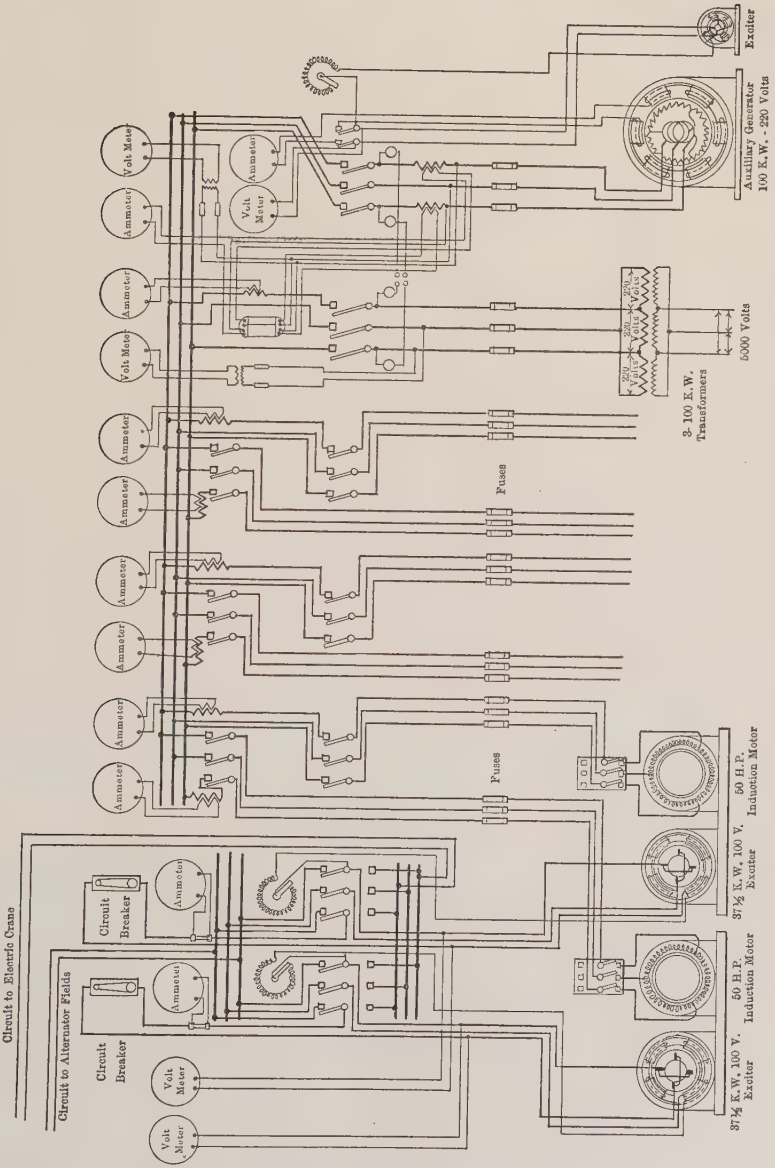


FIG. 11.—WIRING-DIAGRAM OF LOW-TENSION SWITCHBOARD.

5,000 volts down to 220 volts. The current at the lower pressure is delivered to the low-tension board, supplied by the Westinghouse Co., containing the apparatus for controlling the va-

rious circuits in the station; the connections are shown in Fig. 11. One panel is provided for carrying the instruments for controlling the two motor-driven exciter sets, and another panel contains the controlling-apparatus for a small engine-driven, 3-phase, 220-volt alternator of 100 kw. capacity. This latter machine is provided for cases of emergency, when it may be necessary to cut off the supply of power to the station. The small set is constantly under steam, and can be immediately started up and made to supply power to motors driving the auxiliary apparatus in the station until the turbine-generators are able to pick it up. With each change of shift this auxiliary set is brought up to speed and excited, so as to insure proper working when required.

A small "Imperial" Rand air-compressor, driven by a 3-h.p. induction motor, supplies air for blowing-out armatures, etc.

A time-clock, operated by keys inserted into locks placed in various parts of the station, registers the visits of attendants to the apparatus, such as fan-bearings, etc.

An overhead electric crane is provided in the turbine-room of sufficient capacity to lift the generator-armatures, which are the heaviest single pieces, weighing about 20 tons.

IX. Operation of Plant.

The operation of the plant has been most successful, and considering the many features new to the locality, such as mechanical stokers, steam-turbines and polyphase electric work at 5,000 volts pressure, it has shown most satisfactory results.

The following log of coal-consumption is given to show the results obtained under ordinary working-conditions, the figures having been taken from the regular weekly station-report for seven days, from midnight, Sunday, April 2, 1905, to midnight, Sunday, April 9, 1905.

Coal consumed (Glencoe pea), 548,200 lb.; ash removed, 86,700 lb. (Percentage of ash, 15.8.) Kw-hr. generated, 154,840; kw-hr. distributed, 131,314; pounds of coal per kw-hr. generated, 3.54; pounds of coal per kw-hr. distributed, 4.18.

The coal and ash were measured by 20-cu. ft. car-loads, average cars being weighed each day.

A small laboratory for calibrating instruments, making

analyses of coal, and running continuous checks on flue-gas, was built adjacent to the power-station.

The operating staff required in the station is as follows:

1 superintendent, ...	$170.10 per month.
3 turbine-drivers, 8 hr. each per day, responsible for turbines and condensing-plant, at	$4.86 each per day.
3 oilers, working 8 hr. each per day, oiling on turbines and condensing-plant, at	$4.00 each per day.
3 switchboard-attendants, working 8 hr. each per day, at	$4.00 each per day.
2 head boiler-men, working 12 hr. each per day, at .	$4.86 each per day.
2 helpers, looking after coal, ashes, etc., working 12 hr. each per day, at	$4.00 each per day.
2 mechanics, on repairs and maintenance, at . . .	$4.86 each per day.
2 general helpers, clearing up, boiler-cleaning, etc., at .	$3.20 each per day.
16 natives, tipping ashes, shoveling coal, etc., at . .	$1.00 each per day.

South Africa is peculiar in its ideas of dignity of labor. A man who, in America, England or on shipboard, is quite willing to trundle an ash-barrow, scrub a deck or perform other menial service, becomes most unwilling to do the same work in South Africa. He will "boss up" the job—that is, supervise natives—but will not do the work himself.

This view is quite natural when the conditions are understood, generations of custom having relegated to the native certain classes of work which are considered to lie beneath the plane of a white man's calling. Hence the necessity of providing both white and native labor to perform work which, in America, would be performed by white labor alone.

X. Cost-Keeping.

An accurate account of costs is kept of charges in connection with the power-station, and the various departments in the company purchase power at cost price, nothing being added for capital charges. Table I. shows the style of cost-sheet, in which each item has an account number (Coal-handling plant 801, Ash-handling 802, etc.), these numbers being a part of the general cost-keeping system of the De Beers Co.

The various items are arranged as shown, and all charges for work done in the shops of the company are entered against the item for which the work was done.

The charges for attendance, water, fuel, and stores are not entered against individual items, but are lumped against groups

TABLE I.—*Cost-Sheet at*

Working Expenses—Central Power-Station, For.....190...

	Workshop Charges.							Attendance.			Water.		Fuel.		
	Carpenter Shop.	Machine Shop.	Blacksmith Shop.	Boiler Shop.	Electrical Shop.	Painters.	Total.	White Wages.	Native Wages.	Total.	Vaal River.	Dams.	Coal.	Wood.	Workshop Stores and Sidings.
801 Coal-handling plant															
802 Ash handling															
803															
804															
805															
806 Feed-pumps															
807 Filters															
808 Economizers															
809 Feed-piping															
810 Furnaces															
811 Stokers															
812 Boilers															
813 Flues															
814 Draught fans															
815 Steam piping															
816 Blow-down piping															
817															
818															
819															
820															
821 Turbines															
822 Oil system															
823 Generators															
824 Exciters															
825 Switchboards															
826 Auxiliary set															
827 Compressor															
828 Crane															
829															
830															
831															
832															
833 Exhaust piping															
834 Condensers															
835 Air-pumps															
836 Circulating-pump															
837 Make-up pumps															
838 Intake															
839															
840															
841															
842 Station lights															
843 Buildings															
844 5,000 volt cables															
845															
846															
Totals															

Kilowatt hours generated in Power-Station.....Cost per unit.....

Kilowatt hours consumed in Power-Station.....

Kilowatt hours delivered from Power-Station.....Cost per unit.....

(Kilowatt hours received by Company's Works exclusive of Town Lighting, Trams, etc.) .....Cost per unit.....Adjusted.

*the Power-Plant.**

Distribution of Power to Sub-Stations.

Stores.	Stables.		Grand Total.		Kilowatt Hours Delivered from C. P. S. to Sub-Stations. No.	Location.	Kilowatt Hours Received from C. P. S. by Sub-Stations. Location.	Cost.
.....	.....	.....	.....	.....	1	Combined haulage.....	De Beers (surface).....	.....
.....	.....	.....	.....	.....	8	Bultfontein }	" (underground)	.....
.....	.....	.....	.....	.....	10	Dutoitspan }	Pulsator	.....
.....	.....	.....	.....	.....	9	Wesselton.....	Workshops	.....
.....	.....	.....	.....	.....	13	Dutoitspan, No. 2 Machine.....	Kimberley (surface).....	.....
.....	.....	.....	.....	.....	.....	.....	" (underground)	.....
.....	.....	.....	.....	.....	.....	.....	No. 4 washing machine...	.....
.....	.....	.....	.....	.....	.....	.....	Bultfontein (Company Work, surface)	.....
.....	.....	.....	.....	.....	.....	.....	" (underground)	.....
.....	.....	.....	.....	.....	.....	.....	Wesselton (surface)	.....
.....	.....	.....	.....	.....	.....	.....	" (underground)	.....
.....	.....	.....	.....	.....	.....	.....	" (wash. machine)	.....
.....	.....	.....	.....	.....	.....	.....	Dutoitspan (surface).....	.....
.....	.....	.....	.....	.....	.....	.....	" (underground)...	.....
.....	.....	.....	.....	.....	.....	.....	" (open mine).....	.....
.....	.....	.....	.....	.....	.....	.....	" (pan pump).....	.....
.....	.....	.....	.....	.....	.....	.....	Dutoitspan No. 2 Machine)	.....
.....	.....	.....	.....	.....	.....	.....	Kenilworth	.....
.....	.....	.....	.....	.....	.....	.....	No. 3 De Beers	.....
.....	.....	.....	.....	.....	.....	.....	.....	.....
.....	.....	.....	.....	.....	.....	.....	.....	.....
.....	.....	.....	.....	.....	.....	.....	.....	.....
.....	.....	.....	.....	.....	.....	.....	.....	.....
.....	.....	.....	.....	.....	.....	.....	(Town Lighting & Trams Beaconsfield and Ky.)	.....
.....	.....	.....	.....	.....	.....	.....	.....	.....
.....	.....	.....	.....	.....	.....	.....	Total units received.....	.....
.....	.....	.....	.....	.....	.....	.....	Transmission loss.....	.....
.....	.....	.....	.....	.....	.....	.....	.....	.....
.....	.....	.....	.....	.....	.....	.....Totals.....	Totals.....	.....

$$\frac{\text{Total cost of power generated}}{\text{Total units received}} = \text{..... Cost per unit received.}$$

	Profit	Loss
Balance from Kimberley town lighting.....	.....	.....
Balance from Beaconsfield town lighting.....	.....	.....
Balance from Alexandersfontein Trams.....	.....	.....
Balance from Victoria Tramways.....	.....	.....
Net balance to be absorbed.....	.....	.....
£	.....	.....

	Grand Total.
Locomotives.....	.....
Power-station office.....	.....
Salaries.....	.....
Total cost of power generated.....	.....
Balance from town lighting, trams, etc.....	.....
Adjusted total cost of power.....	.....

(* Size of original blank, 16 by 31 in.)

of items, there being five such groups: 1, coal- and ash-handling; 2, steam-generating plant; 3, steam-using plant; 4, condensing-plant; 5, buildings.

The totals are combined, and to this sum are added: locomotives for bringing supplies to plant and taking men to and from work; clerk, stationery, etc., in power-station office; salaries, including superintendence, and a portion of the general management-charges of the entire company, which gives a grand total of the entire cost of power generated.

Certain portions of the power are sold to the towns of Kimberley and Beaconsfield and to trolley-companies operating in them, the profits from such sales being deducted from the generating-costs, and a new "adjusted cost" thus obtained. This adjusted cost, divided by the units received at the various substations, gives the cost per unit chargeable to the mining operations of the company.

The following data, per kw. of capacity of output of station, are of interest:

	Sq. Ft.
Grate area,	0.158
Boiler heating-surface,	12.00
Superheater surface,	1.167
Economizer surface,	2.15
Total heating-surface,	15.317
Condenser surface,	4.25
Area occupied by steam-raising plant, boilers, economizers, flues, stacks, pumps, etc.,	3.03
Area occupied by turbines, generators, switchboards, exciters, condensers, etc.,	1.8
	Cu. Ft.
Hot-well volume,	0.04
Filtered-water storage,	0.14

Present Mining Conditions on the Rand.

BY THOMAS H. LEGGETT, NEW YORK, N. Y.

(New York Meeting, February, 1908.)

In speaking of the mining and economic conditions prevailing at the present time on the Rand, it is not my intention to go into the details of the mining practice, since this has been already well described both in the *Transactions*[1] and elsewhere,[2] but rather to take a more comprehensive view of the industry and its surroundings as a whole, marking its general development and the lines of its future progress.

In a paper entitled Deep-Level Shafts on the Witwatersrand, with Remarks on a Method of Working the Greatest Number of Deep-Level Mines with the Fewest Possible Shafts,[3] presented to this Institute in August, 1900, I drew attention to the great cost of sinking the very deep vertical shafts that were then being seriously proposed as necessary for the development of the deeper areas, pointing out the strong advisability of working this ground by means of inclines to be sunk from the existing shafts of from 3,000 to 4,000 ft. in vertical depth.

In this paper it is stated (p. 975):

"In short, practice upon the Rand now tends towards fewer vertical shafts and more underground inclines, especially for the very deep mining about to be undertaken."

In consequence, the 6,000-ft. vertical shafts, at one time so much discussed, have never been started, and they do not now seem likely ever to be.

The necessity of placing all deep shafts further apart in order to make them control and exhaust larger areas was also dwelt upon. This economy was vitally necessary for both

[1] Gold-Mining in the Transvaal, South Africa, John Hays Hammond, xxxi., 817 to 855 (1901).

[2] *Engineering and Mining Journal*, vol. lxxxv., No. 1, p. 53 (Jan. 4, 1908).

[3] *Trans.*, xxx., 947 to 987 (1900).

financial and technical reasons, very few of the deep shafts being operated to anywhere near their full efficiency, and to further this object the advisability of amalgamating many of the deep-level companies was pointed out.

The hard times experienced on the Rand during the past few years, have emphasized these points very strongly, with the result that the very deep vertical shafts are no longer suggested, while the amalgamation of a large number of the deep-level properties has already been consummated and other amalgamations are under way.

There were many deep-level companies formed 8 or 10 years ago with areas of from 200 to 300 claims (from 280 to 420 acres), which were to be developed by means of vertical shafts placed about 2,000 ft. apart. Many of these properties have been amalgamated, so that the areas are now two or three times the above extent, while wherever possible the distance between shafts has been increased to from 3,000 to 4,000 feet.

In some instances, properties of large area are being opened up by a single deep shaft, which is permissible where connections can be made with adjoining properties, so as to give the two exits required by the Transvaal law.

Under these conditions, several years ago, I started a single seven-compartment shaft to be sunk approximately 4,000 ft. to develop a property of 254 claims (about 356 acres), each of the six hoisting-compartments being 5 by 6 ft. inside of timbers, and the pump- and ladder-way 6 by 6.5 ft., and, as stated, several similar single shafts have since been started.

In short, economy in all directions is the order of the day, and necessarily so, since new capital is not easily found for defraying the cost of expensive deep-level development.

An excellent instance of the amalgamation of deep-level properties, already mentioned, is the consolidation of the Knights Central, Ltd., with the South Knights, Ltd., now under way.

The Knights Central has an area of 444 claims (622 acres), a capital of £525,000, and is developed by means of two five-compartment shafts, about 2,000 ft. apart, and about 2,200 ft. deep each. This company, to-day, has 282,000 tons of ore developed, assaying about 8 dwt., over an average width of reef of 4 ft., which is a highly payable grade of ore, but it has only about £50,000 cash in hand, or not nearly enough to build the

necessary 200-stamp mill and cyanide-plant, and to carry on further mine-development.

The South Knights property has an area of about 450 claims lying on the dip of the Knights Central reef or adjoining that property on the south, where its shallowest depth of reef is about 5,000 ft., and it has a paid-up working-capital of about £300,000 cash in hand. This amount of money is quite inadequate for the necessary shaft-sinking, shaft-equipment and the mine-development, to say nothing of the subsequent milling-plant. In the present strenuous times it would be impossible to raise additional capital, as is evidenced by the fact that not a sod has been turned upon the property, although the company has had nearly $1,500,000 cash in hand since its formation several years ago.

The combining of these two properties gives an area of 894 claims (1,250 acres), insures a very long life to the enterprise, and furnishes the necessary funds for bringing the property to the producing stage, since the cash in hand will then amount to £350,000. It also puts the shareholders of the deeper property within measurable distance (probably 12 or 15 months) of receiving a return upon an investment which has already lain fallow for five years, and which, could they succeed in raising the additional capital necessary to bring it to fruition, would still require six or seven years more.

This example is a fair type of the amalgamations recently accomplished and still going on, chiefly between the mines forming the second row of deep-levels (average depth of reef about 2,600 ft.) and the deeper ground beyond, although in some few instances amalgamations of deep-ground with the nearest outcrop companies are also being discussed.

These combinations are in almost all cases advisable from both technical and financial standpoints. The technical reasons were always more or less apparent, the financial ones not so much so until later years.

Financial Conditions.—It is pertinent to touch upon the financial aspects of these consolidations, not only because the mining business, like all others, is governed by them, but also because one of the leading mining journals of the United States has published the statement that the mines of the Witwatersrand were greatly overcapitalized. I believe it is

demonstrable that the majority of these companies are not overcapitalized.

In the consolidation of the two properties above mentioned, the Knights Central issued £375,000 fully paid £1 shares in payment to the South Knights for its entirely undeveloped property and its £300,000 in cash. The South Knights has a capital of £650,000 (in £1 shares), of which but £500,000 is issued, including 150,000 shares working-capital guaranteed at 35s., this latter feature being the cause of the present large amount of cash in hand.

It is evident that the deeper property suffers a large shrinkage of valuation by this consolidation; it is almost invariably the case that the company owning the deeper and generally largely undeveloped property has thus to suffer in these amalgamations.

With respect to this very deep-level ground, the engineer earliest on these fields, Mr. Hennen Jennings, was always very cautious, while Mr. John Hays Hammond uttered a note of warning in 1901, with which all conservative engineers on the Rand agreed, when he said:[4]

"The results of the developments in the deep-level areas have been so satisfactory as to engender a certain recklessness on the part of mining companies owning very deep reefs. The exploitation of certain of these areas is not regarded by conservative engineers as at present justified, in view of the large intervening tracts of undetermined value which separate mines in operation from the site of proposed mining upon these very deep-level areas."

The redeeming feature of the formation of companies on these deeper areas was the invariable provision of a good round sum for a working-capital, as in the case of the South Knights, and such was the faith of the issuing house in the value of these properties that this working-capital was guaranteed at prices usually much above par.

The result has been, however, that by far the major part of this working-capital has had to be paid by the guarantors (not a pleasant position when the same shares can be bought to-day for one-fourth to one-sixth of the guaranteed price), and this liability can never be avoided, not even by the liquidation of the company.

[4] Gold-Mining in the Transvaal, South Africa, *Trans.*, xxxi., 852 (1901).

It speaks well for the character of the financing upon the Rand that none of these deep-levels was floated without this provision of immediate working-capital and in large sums. The consequence is that the deeper companies are in the position to make much better terms when combining with their northern neighbors than otherwise would be the case, while to this amalgamation many of them must inevitably come, for the reasons already given.

With reference to the alleged overcapitalization of the Witwatersrand companies, a record of the dividends paid by 47 of them (all the data that are now readily available) for the year 1907 shows an average of 44 per cent. on the aggregate capital, while certain properties did much better, as, for instance, the Crown Reef, 210, and the Ferreira, 300 per cent. The Rand Mines, Ltd., comprising a group of nine producing deep-level mines, paid 130 per cent. in dividends in 1907, and promises to increase this rate during 1908. According to Mr. Ross Browne, the total expenditure in shafts, development, plant, and equipment by these nine companies, before reaching the producing-stage, averaged £620,000 per company; consequently, a very considerable capitalization is warranted.

The report of the Mines Department of the Transvaal for November, 1907, shows 71 mines "dropping stamps" on the Rand; hence, it is evident from the foregoing that the majority of these mining companies are far from being overcapitalized.

It is undoubtedly true that the stock market has overcapitalized these companies several times, and probably this fact has led to a confusion of ideas on this subject. Doubtless some of the deep-level mines, more especially the developing ones, to-day seem to be overcapitalized, but they did not so appear at the time of their flotation, else the working-capital would never have been so cheerfully guaranteed at a high premium. Moreover, it is far from certain that they will be so regarded a few years from now, when further development and improved economic conditions bring their welcome changes.

About five years ago I examined a developing deep-level mine that showed only 40 per cent. of 500,000 tons of ore in sight to be payable. To-day this property is making profits of £60,000 per annum and steadily improving, while it is unques-

tionably crushing much of the ore that five years ago was unpayable.

Working-Costs.—As a mining-camp grows older the working-costs almost invariably decrease, providing the camp maintains a healthful activity with advancing years, and this has been the case on the Witwatersrand, the result being as follows:

1898, average working-costs of 65 companies, . . .	25s. 1.3d.
1899, average working-costs of 42[a] companies, . .	25s. 2.7d.
1906, average working-costs of 58 companies, . . .	22s. 1.0d.
1907, average working-costs of 56[b] companies, . .	20s. 8.0d.

[a] The Boer war broke out in October, hence the records are incomplete.

[b] Two less than in 1906, due to exhaustion of the Bonanza mine and incomplete records from one other mine.

These costs include mining, development, crushing and sorting, milling, cyaniding, maintenance, and general expenses, but they do not cover depreciation and amortization, these items being more properly dealt with by the Directors at the end of the year. These results show the very material decrease of 4s. 6d. per ton since 1899, and are therefore approaching now to the 6s. reduction predicted by Mr. John Hays Hammond in 1901, but it has taken time to attain this result, as I then pointed out it would do, and it has not been the result of any change in government. A comparison of the costs in 1907 with those of 1906 shows a decrease of 1s. 5d., or 34 cents, per ton, due chiefly to decreased wages and increased efficiency of both white and colored labor, including the Chinese in the latter category, though increased crushing-capacity through the use of heavier stamps (up to 1,670 lb. per stamp) and regrinding in tube-mills have also aided.

In 1906, 58 companies mined and milled 13,065,624 tons of ore at a total cost of £14,411,219, while in 1907, 56 companies mined and milled 14,861,234 tons at a total cost of £15,351,749, the increase of 1,795,610 tons costing only £940,530.

Most of these economies were attained during the latter half of 1907, after the white-miners' strike, and some mines made startling reductions, as, for instance, the Robinson, which reported costs of 14s. 9d., and the Glencairn, of 15s. 1d. per ton for November. It should be noted, however, that neither of these mines is doing much development-work, having reached the boundaries of their properties with the underground workings.

Such strenuous and successful efforts are now being made to reduce still more the working-costs on the Rand, that I think it safe to anticipate another large decrease for the year 1908.

Output and Dividends.—The total output of the Witwatersrand district for 1907 was 6,220,227 oz. (213.27 tons) of fine gold, equal to an average of 17.75 tons of gold per month. The present yield exceeds 19 tons of gold per month. The value of the output is approximately £26,436,000 ($128,572,000), an average of $10,714,300 per month for the year, not including the "outside" districts of the Transvaal, which are producing in addition about 20,000 oz. of gold per month.

The returns available to date from 56 companies show a total yield during 1907 of £25,154,591, which is equal to 33s. 10.2d. per ton, and a total profit of £9,805,906, equal to 13s. 2.3d. per ton.

The yield per ton for the entire district is practically 34s., or $8.25, which is about 6s., or about $1.50, less than the average return of 10 years ago. On the other hand, working-costs have been reduced 4s. 6d., and are still decreasing, while nearly three times the amount of ore is being crushed as in 1897, although the number of producing companies has not been increased 25 per cent. The number of stamps, however, has been more than doubled, 3,567 stamps in 1897 and 8,375 stamps in November, 1907. Small companies have dropped out and larger ones, both outcrop and deep-level, have taken their places, while many outcrop companies have greatly increased their crushing-capacity.

The dividends for 1907, declared up to January 1, 1908, amount to £6,937,187, or £1,202,026 in excess of the dividends for 1906, and these amounts are exclusive of the 10 per cent. profit-tax, from which the Transvaal government should derive an income for 1907 of about £760,000. These dividends represent slightly more than 25 per cent. of the gross output of gold from this district for the past year, certainly an excellent record for any mining-district.

Hatch and Leggett in the paper, An Estimate of the Gold Production and Life of the Main Reef Series, Witwatersrand, down to 6,000 Feet,[5] predicted an ultimate annual yield of

[5] *Transactions of the Institution of Mining and Metallurgy*, vol. xii., p. 44 (1902–3).

£30,000,000. The yield is now within 12 per cent. of that estimate, which it bids fair soon to reach.

The output for the entire Transvaal for 1907 was £27,403,738, an increase of £2,823,741 as compared with 1906; practically all this increase was due to the Rand proper.

Labor.—The paper by Mr. T. Lane Carter presented to the Institute at this meeting gives a full description of the Chinese-labor conditions on the Rand. There is one important point, however, that does not seem to be generally understood—namely, that the introduction of the Chinese into the Transvaal is not on a par with their introduction into Canada, Australia, the United States, or any of the countries that object to their presence. In these latter countries the Chinaman comes immediately into competition with white labor, while in the Transvaal he does not, because there already exist there two planes of labor, the white and the black, and the Chinaman has been rigorously confined to the latter.

In any country where white and black labor work side by side, there are always certain things the white laborer considers *infra dig.*, and he will always relegate these to the black. The white man is always scrupulously careful to keep the black on the lower level, and to not allow him to rise, since that would mean competition. In the Transvaal the white miner has the overseer's position and the black man does most of the hard work. Engine-drivers and mechanics are white men; in fact, skilled work of all kinds has always been done by the white man, and always will be done by him.

The Chinese have simply supplemented the supply of Kaffirs, and have never been in competition with the white labor any more than are the 100,000 or more Kaffirs now at work in the mines in such competition. Nobody has ever objected to these Kaffirs working with, and under the direction of, the white miners, and as there are 6,000,000 of them south of the Zambesi alone, the country can never be made a "white man's country" in the full sense of that term.

It is evident that the feeling against the Chinese, founded as it is upon a valid and practical cause in other countries, has no such basis in the Transvaal, and is there but an empty sentiment; one, however, that was worked to such an extent in England, where the average man knew nothing of the real con-

ditions and supposed the Chinese would eventually oust the Britons, that the Chinese have to be sent home on the expiration of their contracts. This has been going on for some time, so that out of the 53,000 Chinese in the country at the end of 1906, it is estimated that but comparatively few will remain by the end of this year.

The repatriation of the Chinese is a loss to the mining industry of the Rand, since they are more intelligent than the Kaffirs and have become very adept at mining-work. Moreover, they are on a three-year contract, while the Kaffirs seldom stay for more than six months at a time, though they often return.

The question now is, can all the Chinese be promptly replaced by Kaffirs, to whose scarcity after the war the introduction of the coolie is due. On this point divers opinions have been expressed, but thus far the Chinese have been so replaced, in November, 1907, no less than 4,610 Chinese having been sent away and their places filled by Kaffirs. The Transvaal government, acknowledging its responsibility for sending away the Chinese, has put itself on record as desirous of aiding the mining industry in every way in its power to secure Kaffirs, and as they have perhaps more ways of doing this, it seems highly probable that they will succeed. Moreover the depressed condition of business generally throughout South Africa, combined with the shutting-down of the Kimberley mines, has forced large numbers of Kaffirs to seek employment on the Rand.

As to white labor, the miners' wages, heretofore excessive, have been reduced, partly due to the necessity of giving work to the hundreds of unemployed in and around Johannesburg, with a resultant benefit to all concerned, since the efficiency of this class of labor has been much improved.

Before the strike the white miners' pay was about £1 10s. per fathom stoped, based on the work of the poorest miner, with the result that capable contractors made enormous profits; today contract-prices are reduced, and, what is still more essential, are based upon the capacity of the best miners.

Annual and Quarterly Reports.—The mining companies on the Witwatersrand publish such full and complete reports of their operations, that they can well serve as a model in this respect to the rest of the world. They comprise reports of the board of directors, the consulting engineer and the manager, and give

very full and complete tables, showing the technical details, costs, and results of the operations, together with full financial particulars, comprising usually a balance-sheet, revenue and expenditure, or profit and loss account, and an appropriation account; but above all they show the tonnage in sight at the end of the year and its value, and describe the mine-development and the appearance of the bottom levels of the mine.

The quarterly reports issued regularly to the shareholders give in the same way the results of the preceding three months' work, yield, costs and profit, together with a short description of the appearance of the mine, latest developments, tonnage of ore in sight, and oftentimes its value.

Many mining companies in London now send monthly postal cards to each stockholder giving the amount of ore treated, the yield, cost and profit for the preceding month, and any important feature of the month's development on the mine.

In this day of reform in corporate management it is to be hoped that many of the American mines, well managed in all other respects, will copy the methods and practice of the English companies.

Outlook.—With regard to the outlook for the mining industry, it is several years since I was in Johannesburg, and it is therefore appropriate to quote the views of those who have more recently visited these gold-fields.

Mr. W. Fischer Wilkinson says:[6]

"There is still a considerable margin between the average working costs and the average recovery [*average recovery* $8.25, *average costs* $5 *per ton*—T. H. L.], and a still further lowering of the grade may be expected, carrying with it a corresponding increase in the tonnage available for profitable mining. Low-grade ores of a value of 5 dwt. per ton or so, which were formerly considered worthless or of little value, are now becoming a valuable asset, and the prospective lives of the working mines are constantly having to be lengthened to meet the altered conditions. Undeveloped properties or those partially developed, whose prospects with costs at 30s. per ton or thereabouts were not bright, can look forward to a profitable existence now that costs have been in many cases reduced to 20s. or less. The possibility of working low-grade ores profitably has now been amply demonstrated, and there is no danger of the Transvaal's failing to contribute heavily to the world's supply of gold for many years after the exhaustion of the richer mines of the Central Rand."

Evidence given before the Mining Commission in Johannesburg during 1907 was to the effect that 39 of the dividend-

[6] *Engineering and Mining Journal*, vol. lxxxv., No. 1, p. 53 (Jan. 4, 1908).

paying companies had an average life of 12 years and 8 months. Since the life of every mine exhausted to date has been greatly underestimated, sometimes by as much as 50 per cent., and since much lower grade ore will eventually be worked than is at present milled, it is safe to say that the average life of the above 39 properties will be at least 15 to 16 years. It would be interesting to have the estimated average life of a greater number of the properties; but as there were 71 mines that dropped stamps in November, 1907, of which about 50 are paying dividends, and as more deep-level mines will soon join the ranks of the producers, it seems reasonable to expect not only a maintenance of the present output for some years, but even an increase up to £30,000,000 per annum for a part of that time.

As regards development in the deep-level mines, upon which so much depends, although high-grade ore, assaying in the ounces, was struck in the Cinderella Deep at a vertical depth of 4,000 ft., and while the Knights Central (still developing) has large quantities of 8-dwt. ore at a vertical depth of about 2,400 ft., I am informed that other mines show long stretches of 5- and 4-dwt. ore, with only occasional higher-grade reef.

It is therefore imperative that working-costs should be reduced still further, in the near future, before these deep-level mines reach the producing stage; and it is interesting to note that Mr. Ross Browne, who has recently made a two-year study of the conditions on the Rand, gives, as his opinion, that if the white and the Kaffir labor were made more efficient, which he believes can be done, and the cost of supplies lessened, working-costs could then be reduced to 15s. per ton.

This is a most encouraging statement, coming as it does from a specialist in the subject, and made at a time when the working-costs were 22s. per ton.

There is no doubt that costs will be reduced below 20s. during the present year, probably to 19s., perhaps even better than this, for higher efficiency of both white and black labor is now the earnest objective of all the mine-managements, and when one reflects that these two items comprise about 62 per cent. of the total working-costs it is evident what an effect upon these costs the greater efficiency of labor will have. It is also

to be noted that as the efficiency of the miner increases, the cost of supplies decreases; he consumes less or makes a given quantity go further.

Improvements in the metallurgical practice have brought the extraction up until it is now about 93.5 per cent. of the assay-value of the ore, an important matter when one considers that the saving of one grain of gold per ton of ore crushed means, at the present scale of operations on the Rand, an increased yield of $600,000 per annum. Finally, Mr. Wilkinson considers 5-dwt. ore to be now a valuable asset in any Rand mine—all of which goes to show how time is working in favor of all the mines on the Rand, and especially the deep levels.

In more than one deep-level mine the stopes have turned out better than the assay-plans have indicated, and when one reflects that the drives are now from 200 to 300 ft. apart (even 400 ft. in some deep-level mines), and the connecting winzes separated by distances of 400 ft. and more, it is evident how this may happen. Again, later development has shown that poor zones occur in more or less horizontal stretches, so that the bottom levels of a mine may all be in poor ore at the same time, but on pushing down through this zone the development has again disclosed the normal grade of reef.

Hence, it is now thoroughly well recognized on the Rand that deep-level mines must have from 5 to 10 miles of intelligently directed development-work on the reefs themselves before a valid opinion can be formed of the value.

To summarize, we find that the Witwatersrand output has increased about £2,800,000 in 1907 as compared with 1906, while the dividends have increased £1,200,000, and working-costs have decreased 1s. 5d. for the same period, with another heavy reduction practically assured during the present year.

The use of tube-mills and of heavier stamps has increased the stamp-duty about 1 ton, until it is now 5.8 tons per stamp, while the extraction has been increased to about 93.5 per cent.; hence, tailings that a few years ago ran about 1 dwt., or $1 per ton, now assay but half that amount, with the result that lower-grade ore can now be worked than ever before; in fact, one mine is actually in operation at present on a yield of only $4.31 per ton, or, say, 4.5-dwt. ore.

The possibility of a shortage of Kaffir labor by next July or August is to be considered, but this may easily prove to be not as serious as some now consider it.

The Rand has been going through very depressing times, the natural reaction from over-speculation, with the result that everything is viewed there to-day "as through a glass, darkly."

But the hard times have had a very salutary effect; efforts are being concentrated along the proper lines to secure economy in all branches of its great gold-mining industry, and the results of these are bound to tell favorably in the near future.

The Work of the Testing Department of the Watertown Arsenal, in Its Relation to the Metallurgy of Steel.

BY JAMES E. HOWARD, ENGINEER OF TESTS, WATERTOWN, MASS.

(New York Meeting, February, 1908.)

At the request of the Council of the Institute, I have the honor to submit the following remarks upon the Program of Tests under which the current work of the Watertown Arsenal Testing Laboratory is carried on, which comprises, among other items, the tests of steel ingot metal and derivative shapes.

The intended scope of the tests may be illustrated by extracts from the descriptive part of the program as submitted by the Laboratory under date of April 17, 1907, and subsequently approved by the Chief of Ordnance, U. S. A.

Under Item 1, Ingot Metal—Open hearth, acid and basic; Bessemer, acid and basic; crucible; fluid compressed and uncompressed—descriptive remarks were made as follows:

"A thorough, fundamental inquiry into the physical properties of steel necessarily begins with the ingot metal. Investigations, having for their object the improvement of steels, should also begin with the ingot. A knowledge of the ingot structure is essential for judging of the subsequent behavior or appearance of the metal. Under ordinary conditions, so little information is obtained from the ingot in individual cases that practically none of importance is acquired. These remarks have particular reference to the state of physical and structural

soundness of the steel. Questions of chemical composition, segregation, piping and the grosser cavities have frequently been given attention. Defects of continuity of structure, which manifest themselves in the working of the metal, and at the final stages cause distrust in the finished forging, have generally been neglected.

"The range in physical properties in a given composition of metal, as shown in *Tests of Metals*, 1902 and 1903, is very great. Tests are there found which, when the results are plotted on stress-strain diagrams, practically cover all parts of the diagram above the curve of primitive strength of the ingot. That is to say, the physical properties admit, and quite easily, of such modifications that strong metal, weaker metal, large elongation and contraction of area may be introduced at will in the steel."

To the official program was attached a diagram illustrating the range in tensile properties of a 0.20-carbon steel, the results being brought together from data in the reports of *Tests of Metals*, in order to show how wide such a range could be. The program went on to say:

"It is desired to emphasize this feature, since it has a bearing on the question of establishing limits prescribed by specifications governing the strength of steel for acceptance of the metal. This wide range in strength does not suggest, in itself, what combination of properties is most suitable for a given piece of work, but the arbitrary character of specifications in general is made apparent by an exhibit of the kind shown in this diagram, especially when the results of tensile tests are taken in conjunction with the results of repeated stresses.

"Broadly stated, each grade of steel, chemically considered, can have its tensile properties materially modified by ordinary methods of treatment in common shop use, some grades admitting of wider variations than others, but each admitting of substantial modifications, all states of the metal being serviceable for some constructive purpose, although it is obscure what particular treatment is best for a specific use of the steel. So far as known, desired physical properties may be imparted to steel in its final stages by heat or mechanical treatment—that is, during processes remote from its initial period in the ingot, but unsoundness of structure seems incorrectable.

"These remarks are intended to show what features are regarded as of peculiar importance, and which are best determined in the period when the metal is in the ingot.

"The chief feature to be taken into consideration at the time the metal is in the ingot state is believed to be that of structural soundness and continuity. Evidence seems to show that structural discontinuity in the ingot is not corrected in the subsequent forging operations. These defects are obscured, but not effaced. If the direction of the stresses, in the subsequent test, is in the direction of the flow caused by the forging operations, the defects may pass unnoticed. Their presence, however, is made manifest by the application of stresses applied normal to the lines of structural discontinuity. These remarks are extended somewhat at length on account of the grave importance of the subject. The question of 'streaks,' which has occupied attention for the past ten years, is involved in the matters now being discussed. That streaks have their origin in the ingot, and are due to lines of structural unsoundness, is believed to rest on adequate evidence, and therefore a favorable time to judge of the extent and gravity of such defects would seem to be when the metal is in the ingot. Subsequent tests for acceptance, in the forging, from this point of view, should therefore mainly serve to indicate whether the prescribed tensile properties had been imparted to the metal.

"The ingot tests here contemplated are in the direction of establishing the question of structural soundness and continuity, or chiefly in that direction. For general engineering purposes, this feature is of importance. Steel rails, for railway service, may be mentioned as examples in which structural continuity of the metal is important in a high degree.

"Steels made by each of the three processes, open hearth, Bessemer and crucible, should be taken up in the examination of the ingot metal. The composition of the several ingots examined might properly represent ordinary commercial product, and, when practicable to obtain, similar compositions in the open hearth and Bessemer ingots. Fluid compressed and uncompressed ingots should each be examined with reference to their state of physical soundness and continuity of metal.

"In carrying out experiments on ingot metal as well as along other lines of inquiry, the position of the Testing Labora-

tory should be such as to take advantage of the highest development in the art. That is to say, the material and data collected for the purpose of experimental research should be had from sources where the highest skill now prevails in the art of steel-making. In thus obtaining reliable material, made by steel experts and manufacturers of high reputation, there is assurance that proper material is obtained and also, which is equally important, that the test material represents the quality of good commercial product, and the tests thereupon may be accepted as indexical of good commercial work of to-day.

"In the collection of material, the steel works data on the material used and furnace conditions should be obtained, so far as possible. Test material in general should be well described in respect to its history."

Under Item 2—Blooms, Billets, Rolled and Hammered Shapes from Ingots—the program reads as follows:

"The examination of steel, beginning with the ingot, should be followed through each subsequent stage. There are several stages gone through which do not represent periods at which the product ordinarily reaches the market and is made use of commercially; but, notwithstanding this feature, a complete knowledge of the properties at each stage becomes of value and interest.

"Rolled and hammered shapes will include finished products of all kinds. Some of this class will represent material reduced directly from the ingot, in ordinary manufacturing operations. Material under this heading will also comprise that which is experimental only, being worked up from some of the metal which had been examined in the ingot. Lack of structural continuity in the ingot should be carefully followed in the blooms and billets made therefrom.

"The working temperatures, reductions in the rolls or under the hammer, and the number of passes in the rolls should be ascertained. The material comprises that which is worked above atmospheric temperatures, at forging heats and other high temperatures. Since the physical properties are so readily modified by work done at different temperatures, the need of information on the conditions of treatment as a part of the test record is readily seen.

"In addition to questions on the influence of temperature on

the final properties, there is also the question of interest as to how much or how little work at a given temperature is essential to accomplish the attainable changes in physical properties. That is, to define what amount of work at a given temperature is essential to attain maximum tensile strength and the display of those other features which accompany the tensile test.

"In experimental research many details do not permit of pre-arrangement, and therefore it will not be undertaken to describe the exact nature of many of the proposed tests, only remarking that efforts will be made to discover new data on the properties of the materials, which last remark summarizes the method of procedure, as a rule, in experimental work."

In the inauguration of the tests the Laboratory has obtained the assistance of Mr. William R. Webster, consulting engineer, on matters pertaining to the metallurgy of steel and mill-practice, and of Prof. Edgar Marburg, of the University of Pennsylvania, on the tests on structural members. These groups of tests comprise two of the principal lines of inquiry of the current program.

Through the efforts of Messrs. Webster and Marburg, a representative body of leading metallurgists and engineers of the country convened in New York in September last for the purpose of discussing the work to be undertaken by the Watertown Arsenal Testing Laboratory, and indicating the specific tests, the results of which should furnish information along the lines of immediate interest and importance. This meeting of September 24 last, through the kindness of the Secretary, was held in the rooms of the Institute. At this meeting, which was presided over by Dr. C. B. Dudley, the morning session was devoted to the discussion of structural members, while the afternoon session was occupied upon matters pertaining to the metallurgy of steel, with particular reference to the examination of ingot-metal and rolled shapes. Two committees were appointed by the Chairman, one on structural shapes and one on metallurgical matters. The composition of the latter committee is as follows:

Major C. B. Wheeler, Commanding Officer, Watertown Arsenal (*Ex officio*).

Mr. James E. Howard, Engineer of Tests, Watertown Arsenal (*Ex officio*).

Mr. William R. Webster, Consulting Engineer.

Mr. Edgar Marburg, Professor of Civil Engineering, University of Pennsylvania.

Mr. James Christie, Consulting Engineer.

Dr. Charles B. Dudley, Chemist, Pennsylvania Railroad.

Mr. E. P. Kenney, Metallurgical Engineer, Cambria Steel Company.

Mr. J. P. Snow, Bridge Engineer, Boston & Maine Railroad.

Mr. A. A. Stevenson, Superintendent, Standard Steel Works.

Mr. S. M. Vauclain, Baldwin Locomotive Works.

Mr. F. W. Wood, President, Maryland Steel Works.

In order that these tests may be carried out in a broad and comprehensive manner, an earnest request is made that the members of the American Institute of Mining Engineers participate in the discussion of the work of the Laboratory and indicate the specific tests which, in the judgment of the members, will contribute towards advancing the many questions which so broad a matter as the examination of ingots and rolled shapes involves.

Charcoal and Coke as Blast-Furnace Fuels.

BY R. H. SWEETSER, COLUMBUS, OHIO.

(New York Meeting, February, 1908.)

THERE are so many conditions affecting blast-furnace results that it is hard to get satisfactory comparative data on the working of two furnaces, and much more difficult to get comparable results from the use of two entirely different fuels. The several advantages of charcoal over coke as a blast-furnace fuel have doubtless been apparent to many managers; but probably the conditions for comparative tests of the two were never so favorable as during the year 1905, at the works of the Algoma Steel Co., Sault Ste. Marie, Ontario, which comprise two modern blast-furnaces, well equipped, though of comparatively small capacity. For four consecutive months one of these furnaces was run with charcoal and the other with coke. Then the charcoal-furnace ran for a while with part charcoal and part coke; and, still later, with coke only. The ore-mixtures were

about the same in the two furnaces, and, of course, all the climatic conditions were identical.

The dimensions of the two furnaces, and their work with the different fuels, are given in Table I.

TABLE I.—*Dimensions and Working of Furnaces at Sault Ste. Marie.*

	Furnace No. 1.	Furnace No. 2.
Height,	70 ft. 0 in.	80 ft. 0 in.
Bosh diameter,	13 ft. 6 in.	15 ft. 6 in.
Hearth,	8 ft. 6 in.	10 ft. 0 in.
Stock-line,	9 ft. 6 in.	10 ft. 6 in.
Bell,	6 ft. 0 in.	7 ft. 0 in.
Cubic content,	6,119 cu. ft.	8,913 cu. ft.
Number of tuyeres,	9	9
Diameter of tuyeres,	5 in.	5 in.

	Charcoal.	Coke.	Coke.
Maximum day's product,	173 tons.	237 tons.[a]	339 tons.
Maximum week's product,	1,004 tons.	1,453 tons.	2,175 tons.
Maximum month's product,	4,071 tons.	6,131 tons.[b]	8,027 tons.
Date of blowing in,	Mar. 6, 1905.	July 16, 1905.[c]	Oct. 17, 1904.

[a] Feb. 4, 1906. [b] March, 1906. [c] Changed from charcoal and coke to all coke.

Two important facts were demonstrated during the four months' run of No. 1 with charcoal: (1) that a modern blast-furnace of approved form and construction, equipped with up-to-date machinery, and 70 ft. high, could be successfully operated with charcoal as fuel; (2) that, with few exceptions, the equipment and the blast-furnace practice best suited to successful coke-furnaces were also well adapted for the charcoal-furnace.

TABLE II.—*Data of Results of Furnace Operations.*

Furnace No. 1, Charcoal.

1905.	Theoretical Yield. Per Cent.	Actual Yield. Per Cent.	Loss in Yield. Per Cent.	Mesabi Ore. Per Cent.	Cu. ft. Blast per Ton Pig.	Fuel. Lb. per Ton	Quantity of Stone per Ton of Pig. Lb.	Product for Month. Tons.	Average Quantity per 24 Hrs. Tons.	Quantity of Blast per lb. of Fuel. Cu. ft.
April	57.20	56.40	0.80	33.4	91,421	2,120	393	3,722	124.1	43.11
May	58.51	58.30	0.21	33.7	83,899	2,016	308	4,040	130.3	41.60
June	57.73	55.60	2.13	34.0	99,495	2,114	403	3,732	118.9	46.94
Average	57.81	56.76	1.05	33.7	91,605	2,083	368	3,831	124.4	43.88

91,605 cu. ft. of air = 7,338.4 lb. = 3.27 tons of air per ton of pig (charcoal).

Furnace No. 2, Coke.

April	57.30	54.70	2.60	23.5	134,739	2,171	1,036	5,267	205.3	60.17
May	58.79	57.80	0.99	26 9	140,307	2,107	1,037	7,148	230.6	66.52
June	58.16	55.00	3.16	28.3	148,103	2,343	1,050	6,670	227.8	63.20
Average	58.08	55.83	2.25	26.2	141,049	2,207	1,041	6,361	221.2	63.29

141,049 cu. ft. air = 11,283.9 lb. = 5.03 tons of air per ton of pig (coke).

Tables II. and III. give the averages of the results obtained during the three months of the period under consideration. No allowances have been made for lost time, although furnace No. 2 was shut down for several days in April to put on a new tuyere-jacket.

TABLE III.—*Analyses of Pig-Iron and Slag.*

Furnace No. 1.

	Analysis of Pig Iron.				Analysis of Slag.							
1905.	Si. Per Cent.	S. Per Cent.	P. Per Cent.	Mn. Per Cent.	SiO_2. Per Cent.	Al_2O_3. Per Cent.	$SiO_2+Al_2O_3$ total. Per Cent.	Fe. Per Cent.	Mn. Per Cent.	CaO. Per Cent.	MgO. Per Cent.	S. Per Cent.
April	1.22	0.010	0.086	0.52	42.15	11.02	53.17	0.74	0.62	43.00	1.25	0.24
May	1.63	0.014	0.083	0.61	44.49	12.02	56.51	0.87	.....	.....	.....	.....
June	1.53	0.010	0.082	0.51	45.76	9.99	55.75	1.29	.....	.....	.....	.....
Average	1.46	0.011	0.0836	0.546	44.13	11.01	55.14	0.966	0.62	43.00	1.25	0.24

Furnace No. 2.

April	1.48	0.031	0.074	0.49	37.48	11.84	49.32	0.63	0.34	48.10	1.46	1.32
May	1.39	0.027	0.072	0.53	38.12	11.77	49.89	0.62	.....	.....	.....	.....
June	1.50	0.029	0.067	0.54	39.17	10.94	50.11	0.74	.....	.....	.....	.....
Average	1.46	0.029	0.071	0.52	38.26	11.51	49.77	0.663	0.34	48.10	1.46	1.32

By comparing the results, of Tables II. and III., it will be seen that charcoal has the following metallurgical advantages:

1. The furnace consumes considerably less charcoal than coke per ton of pig.

2. Only about one-third as much limestone per ton of pig-iron is required in a charcoal-furnace as in a coke-furnace.

3. The amount of blast required for a charcoal-furnace is only about 65 per cent. of that for a coke-furnace of the same productive capacity.

4. The "critical temperature" in a charcoal-furnace may be lower than that required in a coke-furnace.

These four advantages involve many other smaller ones which affect the cost of equipment and of operating: *i. e.*, less fuel and limestone to handle; less blowing-engine equipment; less steam; less hot-blast stove area for heating the blast; lower blast-pressure on the engines, stoves and connections; less flue-dust carried over by the waste gases, and, consequently, a little higher yield from the same ores; less cooling-protection in the hearth- and bosh-walls; and practically entire freedom from any high-sulphur pig-iron.

It was very evident during the three months of the above

comparison that the charcoal-furnace could not do itself justice on account of the poor quality and the insufficient supply of the charcoal. But even under these adverse conditions the results of three months' work showed the advantages of the charcoal.

Quantity of Fuel Per Ton of Pig.—In comparing the quantity of fuel required per ton of pig, the results given are based on the net tonnage of pig delivered to the Bessemer Department, and the gross tonnage of coke and charcoal delivered to the furnaces, including waste. On account of the frightfully poor quality of the charcoal this waste, at times, amounted to 16 per cent. The results for the period of three months show a consumption per ton of pig-iron of 2,083 lb. of charcoal, as compared with 2,207 lb. of coke.

The work for a coke-furnace was exceptionally good, while for a charcoal-furnace it was very poor. Every pound of coke less than 2,240 lb. per ton of pig will be an almost unlooked-for gain at the "Soo" unless the yield of the ores is much higher than at present. During 1905 the average quantity of coke per ton of pig-iron required by the furnaces in the Mahoning and Shenango valleys, exclusive of those of the U. S. Steel Corporation, was 2,311 lb. Including these latter furnaces, which make a total of 89 furnaces, it was 2,234 lb.

The quantity of charcoal consumed per ton of pig—viz., 2,083 lb.—was excessive. If the quality had been good, the consumption would have been nearer 1,800 lb. Some furnaces using good charcoal have made iron with 1,600 lb. of charcoal per ton of pig. It is safe to count on a saving of 400 lb. of fuel in favor of charcoal.

On account of the practical absence of mineral matter in the charcoal, there is no fuel consumed to slag the ash, nor is there need of limestone to flux it. Moreover, the entire absence of sulphur in the charcoal makes it possible to have a very acid slag, which is consequently much less refractory than the slag necessary for the same ores using coke as fuel.

Limestone Per Ton of Pig.—A comparison of the analyses given in Table III. shows that much less lime is required in the charcoal-iron slag than in the coke-iron slag. The chief reason for this is that there is scarcely any sulphur to be carried off in the charcoal slag. In addition to this advantage, there is also a greater gain by not having to flux the ash. But the practice

with charcoal at the "Soo" differed somewhat from that in northern Michigan, where much less limestone is used. At the suggestion of Mr. Wm. Wilkins, of Ashland, Wis., less limestone was tried, but the results were less satisfactory. A slag containing SiO_2, 40.78; Al_2O_3, 13.20; and Fe, 0.35 per cent. was all right, and the furnace worked smoothly on it under a blast-temperature of 1,200° F. A slag with SiO_2, 38.74; Al_2O_3, 12.35; and Fe, 0.20 per cent. was a little "limey," and one with SiO_2, 37.12; Al_2O_3, 11.54; and Fe, 0.25 per cent. was too "limey." A slag with SiO_2, 55.52; Al_2O_3, 9.13; and Fe, 0.90 per cent. worked very badly.

In a charcoal-furnace there is the same chance as in a coke-furnace to have the mixture too "dry." The volume of slag must be sufficient to make the furnace work smoothly, and eliminate troubles arising from a cold, stiff-running slag.

One charcoal-furnace manager wrote me that he used only 175 lb. of limestone per ton of pig-iron, with ores carrying about 6 per cent. of silica.

The small amount of limestone required in a charcoal-furnace gives less weight of material to handle and less bulk in the furnace itself, which reduces the cost of stocking and charging. It also gives less slag to handle, and reduces the cost of switching and of repairs to the cinder-ladle.

Air Per Ton of Pig.—One of the great advantages of the use of charcoal over coke is the smaller quantity of blast required per ton of pig. A pound of coke requires nearly 50 per cent. more air for its combustion than does a pound of charcoal. This great saving allows cheaper blowing-engines, less boiler-capacity and less hot-blast stove capacity; consequently, the lesser cost for installing and operating engines, boilers, stoves, pumps, steam-lines, blast-mains, gas-mains and connections. This smaller quantity of blast naturally carries much less moisture into the furnace. Moreover, the labor for operating and for repairs is less.

Another advantage of this smaller quantity of blast is the smaller margin of loss, as shown in Table II., by a comparison of the difference between the theoretical and the actual yields. The loss in yield of the charcoal-furnace was only 1.05 as compared with 2.25 per cent. of the coke-furnace. The charcoal-furnace carried 7.5 per cent. more Mesabi ores than the coke-furnace.

The great difference in blast-volume for the two fuels was most clearly demonstrated when the change was made from charcoal to coke. No stop or interval was made, the coke being gradually substituted for charcoal, and the engine-revolutions increased to meet the demand for more air.

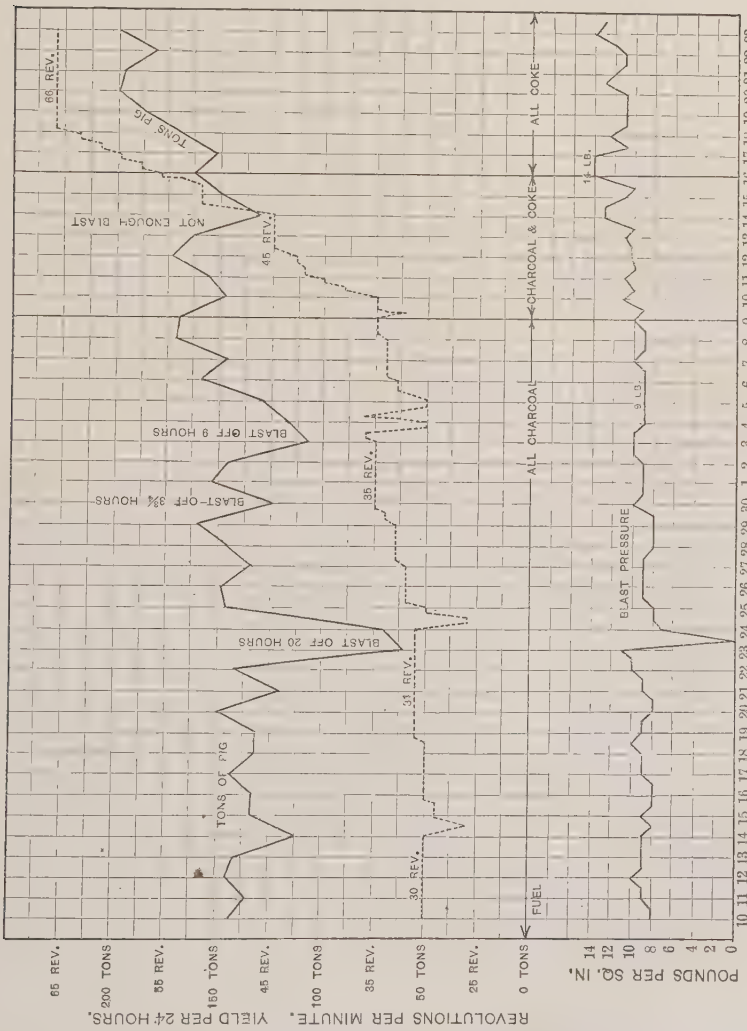


Fig. 1.—Data of Operations of Furnace No. 1, at Sault Ste. Marie, Ont.

The chart, Fig. 1, shows the average blast-pressure, tons of pig-iron per day, and average number of rev. per min. from June 10 to July 23. During the greater part of this period the furnace was running on charcoal, and the quantity of blast blown (and consequently the amount of pig-iron made) was

regulated by the daily supply of charcoal, which was never sufficient to blow the furnace as hard as it should have been blown for most advantageous working. Two abnormal results were brought about by this light blowing: 1, the blast-pressure of from 9 to 10 lb. was higher than it should have been; and 2, the furnace had practically a ring-scaffold, which was gradually worked off later, after coke was used and more blast was blown into the furnace.

During the last week in June, when it was seen that the use of charcoal must be abandoned, a small extra supply was obtained, and the engine-revolutions were increased for a while, which gave an increased product, but there was no increase in the blast-pressure. Trouble with the hoist and a shortage of charcoal on July 3 caused a shut-down of 9 hr. But from July 6 to 10, there was enough charcoal to allow blowing at the rate of from 34 to 35 rev. per min. Under these conditions, the consumption of charcoal, per ton of pig, was 1,725 lb., and the quantity of product rose to 173 tons per day. Then, for a week, a mixture of charcoal and coke was used, the proportion of coke being gradually increased until all the charcoal was replaced, which occurred at the second charge on the day-turn of July 16. Although endeavors were made to increase the quantity of blast fast enough to take care of the coke as it came to the tuyeres, it was not fast enough on the 13th and 14th, and consequently the rate of production dropped back. A more rapid increase in volume of blast brought the expected increase in product. During the transition from charcoal to coke, both fuels could be easily seen at the tuyeres mixed together.

The Critical Temperature.—The construction of the furnace was not changed after supplanting charcoal by coke. But the increase in blast-volume and temperature of the hearth soon began to affect the brick-work of the hearth, the tuyere-circle and bosh-walls. Extra bosh-plates were inserted between the tuyere-coolers, and later much trouble arose from thin walls all around and above the tuyeres. Our experience showed that in a coke-furnace stronger walls and more water-cooling devices are necessary than in a charcoal-furnace.

In a coke-furnace the slag is more refractory than in a charcoal-furnace, and for this reason, in making pig-iron of a given content of silicon, it was necessary to have a hotter hearth in the coke-furnace than in the charcoal-furnace.

The Product.—The average silicon in the pig-iron made during the three months' run was identical in the two furnaces, but the average sulphur in the charcoal-iron was only 0.011 per cent. as compared with 0.029 per cent. Rarely did the sulphur in the charcoal-iron exceed 0.021 and never more than 0.032 per cent. On account of the higher phosphorus-content of the charcoal, the charcoal-iron contained more phosphorus than the coke-iron from the same ores. The manganese-content in both kinds of iron was about the same.

Disadvantages of Charcoal.—There are two serious disadvantages in using charcoal instead of coke, although both can be overcome. The first is the difficulty of getting a sufficient supply of good charcoal. The cord-wood needed for making the daily supply of charcoal for a furnace producing 150 tons of iron per day would make a pile half a mile long. The necessary areas of maple and birch forests are becoming less and less plentiful in the United States, although in Canada there are immense tracts of wooded lands waiting to be cleared for settlers, which contain much of the wood that is better suited for making charcoal than it is for use as lumber.

The second disadvantage is the readiness with which charcoal catches fire, a danger which is not past until the charcoal reaches the tuyeres. Even in the upper part of the furnace, there is a chance of serious trouble from the flaming charcoal if the furnace should make a heavy slip. Furnace No. 1 made many heavy slips on account of dirty charcoal and insufficient blast-volume, and almost always the fine charcoal-dust would ignite as it was thrown out of the top explosion-doors. At times the flames would reach the ground. But all this could be avoided with a sufficient supply of good charcoal from kilns close to the furnace.

It is interesting to note that during the time furnace No. 1 was running on charcoal and making the largest recorded outputs for charcoal-furnaces, there were also in blast in Canada what are probably the smallest charcoal blast-furnaces in active operation in the world. In Drummondville, Quebec, two small stone-stack charcoal-furnaces of the old French type, built in 1880 and 1881, were operated in the most primitive style and gave a total daily output of 3.5 tons of pig-iron per furnace. The dimensions of one of these furnaces are: 31 ft. height, 10 ft. bosh and 3 ft. 8 in. sq. hearth; and of the other, 32 ft. height, 9 ft. bosh and 3 ft. 8 in. sq. hearth.

The Coal-Briquette Plant at Bankhead, Alberta, Canada.

BY EDWARD W. PARKER, WASHINGTON, D. C.

(New York Meeting, February, 1908.)

THIS plant was built in 1907 at the Bankhead mines to manufacture briquettes by the Zwoyer process under license from the Zwoyer Fuel Co., of New York, N. Y.

The building was constructed to contain two units, each of a capacity of 10 tons per hr.; one unit was installed at the time the building was constructed, and the second unit is now being built.

The coal used is an anthracite, which is more friable than that of Pennsylvania, and, as a consequence, a larger percentage of dust or waste is produced, which was formerly thrown upon the slack-pile as waste, but is now passed to the "dust-bin" in the breaker, from which it is conveyed to the briquette-plant.

An average analysis of the coal, published by Lewis Stockett and B. R. Warden,[1] is as follows:

	Per Cent.
Moisture,	0.50
Volatile,	8.00
Fixed carbon,	83.50
Ash,	8.00
Total,	100.00
Sulphur,	0.40 per cent.
Specific gravity,	1.40
Color of ash,	white
Heat value,	14,000 B.t.u.

The plant is run under two 12-hr. shifts, the force consisting of 1 superintendent for both shifts, 1 engineer for each shift, 1 briquetter for each shift, 1 helper for each shift, 1 laborer cleaning up for each shift. In the pitch-melting house, 2 men are on duty for one shift.

The steam for the engine is furnished from a central boiler-plant, which also supplies steam for the breaker, the electric light and power-plant, machinery-shop, and boiler-shop.

During March, April, May, and half of June, in 1907, the

[1] *Journal of the Canadian Mining Institute*, vol. ix., p. 261 (1906).

plant was operated on part time, due to the reduction of the boiler-capacity on account of the formation of heavy scale on the boiler-tubes and sheets, and to the miners' strike. About the middle of June an additional boiler was installed, and the water from a spring supplied by the melting snow was used, which enabled the briquette-plant to be operated continuously from Monday morning until Sunday morning. Any loss of time at present will be from failure to obtain cars for loading the briquettes. The output during six months of 1907 was: June, 4,882; July (459 hr.), 4,593; August (499 hr.), 7,054; September, (420 hr.), 5,432; October (570 hr.), 7,668; November (558 hr.), 8,147, the total for the year being 43,703 tons.

During October, notwithstanding the loss of 5 days of operation (4 days on account of car-shortage and 1 day occupied in installing a pitch-tank for the second unit), the production amounted to 7,668 tons in a total running-time of 570 hr., and in November the total output was 8,147 tons and the running-time 558 hr. The average run per hour in October was 13.45 tons, while in November it was 14.6 tons.

The latest information regarding the operations of the Bankhead plant was received about Jan. 1, 1908, to the effect that a second press had been shipped from New York early in December, and as soon as this press was in place, so that a sufficient output could be obtained, one division of the Canadian Pacific Railroad would burn nothing but briquettes on its locomotives. This indicates that the experiments up to the first of the year had been quite satisfactory, otherwise the railroad company would not have gone to the expense and trouble of drafting its locomotives to burn briquettes exclusively. When the additional unit is in operation, the plant will probably produce from 15,000 to 16,000 tons of briquettes per month.

Domestic consumers seem to be well pleased with the product, and there is a large demand for briquettes, which sell for $4 per ton at the plant.

Fig. 1 is a general view of the Bankhead Mines buildings, including the briquette-plant at the right of the breaker. The boiler-plant is located on the right between the briquette-plant and the electric light and power-plant. The slack-pile is shown on the low ground at the left of the breaker.

Figs. 2 and 3 show the relative location of the breaker and

the briquette-plant, the latter consisting of a pitch-melting house, and the main building, which includes dust-bin, machinery-house, and cooling-house.

The pitch-melting house, about 23 by 47 ft. in area, is separated from the main building, and contains two melting-tanks, each 12 by 6 ft. and 8 ft. deep.

The dust-bin, 20 by 30 ft. and 24 ft. deep, is placed at the rear of the machinery-house. This house is 60 by 73 ft. in area, and contains the briquette-machinery, crusher, mixer of six units, pitch-storage tank, 12 by 6 by 8 ft., double-cylinder engine, 14- by 20-in. stroke, two dust-elevators, and briquette-press. The cooling-house, 31 by 129 ft. in area, contains the cooling-table and machinery for driving the briquette-conveyor.

Fig. 4 is an interior view of the machinery-house, and shows the press at the left, back of which is the engine; the conveyor from No. 6 mixer to the press; the mixers in the center and back; the pitch-storage tank at the right, and bins forming the end of the building.

The dust from the operations of the breaker is conveyed to a dust-bin at one end of the breaker, from which it is taken by a scraper-conveyor to the dust-bin in the briquette-plant. The dust is removed from the bins and conveyed to the crusher by a drag-chain. It then passes through the crusher and is elevated to the mixers, where it is heated by means of the hot gases from the mixer-furnace, which enter the mixers through flues in the sides. The pitch is introduced by means of an atomizer while the dust is passing through the mixers. As soon as the mixture reaches the end of mixer No. 6, it is conveyed by an elevator to the press and briquetted. The briquettes are carried by a belt-conveyor to the distributor over the cooling-table, and are carried back and forth the length of the table seven and three-quarter times, finally being dropped into the briquette-conveyor, which carries them to the briquette-bins in the breaker-house. From these bins the briquettes are loaded by means of chutes into gondola cars, and by means of a Victor box-car loader into box-cars.

All of the material is handled by machinery, which gives a continuous operation from the time the dust leaves the breaker until it is returned to the breaker-house in the form of briquettes.

A, Slack-pile. B, Breaker. C, Briquette-plant. D, Boiler-plant. E, Electric-plant.

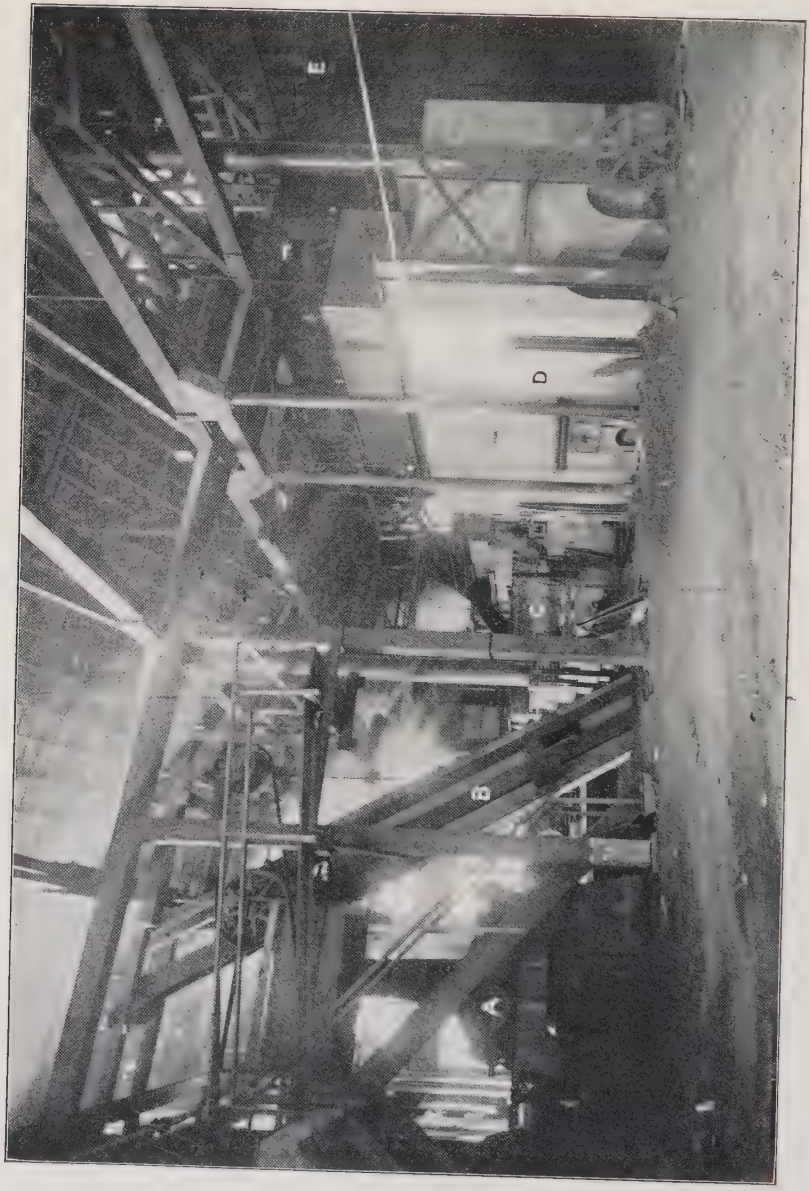
FIG. 1.—GENERAL VIEW OF THE BRIQUETTE-PLANT AT BANKHEAD, CANADA.

A, Cooling-house. B, Dust-bin. C. Dust-conveyor. D, Briquette-conveyor. E, Box-car loader. F, Breaker.

FIG. 2.—VIEW OF BREAKER, CONVEYORS, DUST-BIN, AND COOLING-HOUSE, AT BANKHEAD, CANADA.

A, Pitch-melting house. B, Dust-bin. C, Briquette-machinery house. D, Breaker. E, Cooling-house. F, Slack-pile.

FIG. 3.—VIEW OF BREAKER, BRIQUETTE-BUILDINGS, AND COOLING-HOUSE, BANKHEAD, CANADA.



A, Press. B, Elevator to press. C, Mixers. D, Pitch-storage tank. E, Dust-bin.

FIG. 4.—INTERIOR OF BRIQUETTE-MACHINERY HOUSE, AT BANKHEAD, CANADA.

Calculation of Mine-Values.

BY R. B. BRINSMADE, PLATTEVILLE, WIS.

(New York Meeting, February, 1908.)

THE following is an attempt to form a formula by which a mine can be quickly evaluated, after all pertinent physical data have been collected from observations on the ground by a competent mining engineer.

ASSUMPTIONS.

Let
- G = price to be paid for the mining property.
- M = cost of developing the mining property to yield y tons of ore daily.
- P = cost of suitable plant to treat y tons of ore daily.
- p = value of said plant when the mine has been exhausted.
- C = total fixed capital investment.
- W = working capital investment.

Then let C + W = total capital investment.
- y = required yield of ore daily in tons.
- Y = yield of ore yearly in tons.
- d = number of producing days per year.
- u = average operating profit per ton of ore.
- R = rate of interest to be earned on total investment of (C + W).
- r = rate of interest to be earned on sinking fund annuity.
- v = tons of positive ore available in mine.
- x = tons of probable ore available in mine.
- z = tons of possible ore available in mine.
- m = fractional factor to change probable to positive ore.
- n = fractional factor to change possible to positive ore.
- Q = tons of total ore available in mine.
- t = time in years to exhaust mine at rate Y.

Let b = time in years for mine to reach production of y tons.

A = annuity to be paid to sinking-fund to equal C at end of t years at r, compound interest.

a = annuity to be paid to sinking-fund to equal C at end of t years at r, simple interest.

Derivation of Formulas.

We have directly from the foregoing assumptions the following equations:[1]

$$C = G + M + P \quad (1).$$

$$Y = d\,y \quad (2).$$

$$Q = v + mx + nz \quad (3).$$

$$Q = t\,Y \quad (4).$$

[1] In a contribution to the *Engineering and Mining Journal*, of March 14, 1903 (vol. lxxv., No. 11, p. 401), Mr. G. E. Collins has given a method of measuring v, x and z, and of determining safe values for m and n.

In this article he presents, as a practical example, a longitudinal section (Fig. 1) of a fissure-vein, the ore-reserves of which he classifies as "positive" (proved on three sides), "probable" (proved on two sides), and "possible" (proved on one side). Upon the case thus supposed, he makes the following remarks, which I deem worthy of preservation as generally applicable and instructive:

"Most of us have known, many of us have experienced, cases where blocks of ore so exposed [even] on four sides have been found to enclose a large barren patch in the center. However rigid a rule we may adopt, we cannot arrive at certainty.

"The amount of weight to be attached to exposure on four sides, on three, on two, or on one only, varies with the conditions of each particular case. . . . Who, after examining the section [Fig. 1], and bearing in mind the phenomenal lateral extension of the ore-body shown, can doubt, that far more reliance can be placed on blocks marked in the section as "positive ore," even although some of them are exposed on three sides only, than on many blocks opened up on four sides in veins where the ore-bodies are of the bunchy and erratic type which we must recognize to be, after all, by far the most frequently encountered? Even the "possible ore" of Fig. 1—ground where the ore is proved to exist on one side only—can be depended on to a far greater extent than usual.

"My contention is that the amount of evidence to be required when making estimates of ore tonnage, and the number and nature of classes into which estimates should be divided, must depend on our general conclusions as to the nature and permanence of the ore-bodies in the mines under consideration, the distance between the workings, etc. I do not think that any rules can be made which will lessen the necessity for dependence on the examining engineer's individual judgment; and I distrust all cast-iron classifications, which do not allow for the infinite complexity of natural conditions. The only useful general rule is that no estimate of tonnage should be made unless accompanied by sketches indicating the basis on which it rests."

Then since we must balance the two debits, of the interest to be paid during (t + b) years and the sinking-fund annuity to be paid during t years, against the two credits, of the operating profit from the total available ore and the selling value of the abandoned plant, we have:

$$(t + b)(C + W)R + tA = tYu + p \qquad (5).$$

To determine A, we can substitute in that algebraic formula for the annuity which involves the rate of compound interest, the number of annuity payments and the capital to be refunded and get:

$$A = \frac{Cr}{(1 + r)^t - 1} \qquad (6).$$

Substituting in (5) this value of A, we have:

$$(t + b)(C + W)R + \frac{tCr}{(1 + r)^t - 1} = tYu + p \qquad (7).$$

With the observed data, and equations (1), (2), (3), and (7), we can proceed to evaluate the mine. Equation (7) includes nine factors, any one of which, except t, can be determined by solving a simple equation. When t is unknown (as is commonly the case), it is difficult to solve the equation by common algebraic methods. To obviate this difficulty, it may be assumed that simple, instead of compound, interest is to be earned on the sinking-fund annuity; and the resulting difference in the result will act as a safeguard on the side of the mine-buyer. Additional safeguards for the buyer will be the assumption, in equation (7), that the factors C and W are invested at once, whereas their expenditure usually occupies a considerable period.

With simple interest, the sinking-fund annuity (a) becomes the first term of an arithmetical progression, of which the common difference is the annual interest (ar) gained on the annuity, the sum is the capital (C) to be repaid, and the number of terms is (t) the period of years. Then, by substitution in that algebraic formula for the sum which involves the first term, the common difference and the number of terms, we have:

$$C = \frac{t}{2}\left(2a + (t-1)\,ar\right) \text{ or } a = \frac{2\,C}{t\left(2 + r\,(t-1)\right)} \qquad (8).$$

Substituting this value of a for A in (5), we have:

$$(t+b)\,(C+W)\,R + \frac{2\,C}{2 + r\,(t-1)} = tYu + p \qquad (9).$$

To determine from equation (9) the unknown factor t, will involve only the solution of a quadratic equation; and this is most conveniently performed after the substitution of the numerical values of the other factors; since the direct solution for t, of (9) as a literal equation, is very lengthy.

If no interest be earned on the annuity payments, the second term of equation (9) will become equal to C, and we shall have:

$$(t+b)\,(C+W)\,R + C = tYu + p \qquad (10).$$

Since equation (10) involves only first powers, it can be used as a quick check on the approximate accuracy of the solution of equation (9) for t.

Practical Examples.

I.

On a certain property in the Southwest, examined by me, it was required to ascertain the minimum available ore that must be found by prospecting operations to warrant the capital expenditure required to inaugurate production on a given scale. The unknown factors were hence Q and t, and the known were estimated from the collected data to be:

G = \$40,000; M = \$56,000; P = \$39,000; p = \$10,000; W = \$15,000; y = 100 tons; d = 300 days; u = \$3; R = 0.15; r = 0.06; b = 2 years.

From equation (1), $C = 40{,}000 + 56{,}000 + 39{,}000 = \$135{,}000.$

(2), $Y = 300 \times 100 = 30{,}000$ tons.

$$(9),\ (t+2)\,(135{,}000 + 15{,}000)\,0.15 + \frac{2 \times 135{,}000}{2 + 0.06\,(t-1)} = t\,(30{,}000 \times 3) + 10{,}000;\ \text{or}$$

$$27\,t - 14 - \frac{108}{1.94 + 0.06\,t} = 0;\ \text{or}$$

$$27\,t(1.94 + 0.06\,t) - 14\,(1.94 + 0.06\,t) - 108 = 0, \text{ or}$$
$$0.81\,t^2 + 25.77\,t - 67.58 = 0; \text{ whence}$$
$$t = 2.42 \text{ or } -34.24.$$

It is evident that the positive value of 2.42 years is the one desired.

Substituting the values of t and Y in equation (4), we have:

$$Q = 2.42 \times 30{,}000 = 72{,}600,$$

the number of tons of available ore, that should be found by prospecting, to satisfy the conditions of the case.

II.

A consumer, using 5,000 tons of a certain metal yearly, wishes to acquire a mine which would furnish his whole supply. He has found a mine which, by the expenditure, besides the purchase-price, of $200,000 for development and plant, and the provision of $50,000 working capital, would enable him to obtain annually the required supply of metal from 60,000 tons of its ore, at an operating profit of $4 per ton. A year will be required to develop and equip the property, and the available ore will last, at the required rate of production, for 20 years, at the end of which period the plant will be worthless. If interest on the total investment be reckoned at 6 per cent. and on the sinking-fund annuity at 5 per cent., and if, in addition, it is necessary, while working the mine, to make a net saving of 1 cent a pound on the whole metal-supply of the consumer, what price could he afford to pay for the mine?

Here C and G are the unknown factors, and we have:

$M + P = \$200{,}000$; $W = \$50{,}000$; $u = \$4$; $p = 0$; annual saving on metal, $100,000; $Y = 60{,}000$ tons; $R = 0.06$; $r = 0.05$; $t = 20$ years; and $b = 1$ year.

From equation (1), $C = G + 200{,}000$.

Substituting this value in equation (7), and remembering that the sinking-fund annuity is to be increased by the $100,000 of annual saving on metal-supply, we have:

$$(20 + 1)(G + 200{,}000 + 50{,}000)\,0.06 + \frac{20\,(G + 200{,}000)\;0.05}{(1 + 0.05)^{20} - 1}$$
$$+ (20 \times 100{,}000) = (20 \times 60{,}000 \times 4) + 0; \text{ or}$$

$$1.26\,(G + 250{,}000) + \frac{G + 200{,}000}{2.654\text{—}1} + 2{,}000{,}000 = 4{,}800{,}000;\ \text{or}$$

$$2.084\,(G + 250{,}000) + G + 200{,}000 + 1.654\,(2{,}000{,}000) = 1.654\,(4{,}800{,}000);\ \text{or}$$

$$G = \$1{,}264{,}500,$$

the maximum allowable price for the mine.

Conclusions.

The practical value of such calculations as the foregoing may be plausibly questioned, at least as regards all mines other than collieries, for which the quantity of available reserves can be estimated with a degree of confidence and precision not usually attainable in mines of the metallic ores. In reply to this probable criticism, I beg to offer the following observations:

1. There are, in fact, besides collieries, more mines than we commonly realize, the actual reserves of which can be measured, and the probable or potential reserves estimated. Among such I might instance many quarries, massive ore-bodies already explored by boring, etc. To all such cases, mathematical formulas of valuation are directly applicable.

2. With regard to the very large number of metal-mines, in which v, the certain ore-reserve, is relatively small, while x, the probable, and z, the possible reserve, are so uncertain as to make the determination of m and n, as the moduli representing reasonable expectation, merely a function of the temperament of the observer, I beg to say that, even in such cases, the observer himself may, very likely, be steadied in his judgment, and aided to form prudent conclusions, by such calculations as will clearly show him to what extent his hopes, fears, and guesses enter into his opinions. It may seem absurd to employ mathematical methods in the discussion of data so largely indefinite; but the quantitative determination, even of our ignorance, is a recognized application of mathematics; and "the probable error" has a value of its own, not less real, though it be less authoritative, than the rigorously demonstrated certainty. In other words, I venture to believe that the discussion, by exact methods, even of more or less uncertain data, is a valuable check upon the hasty sentimental or temperamental general impressions which often claim the authority of conclu-

sions. Such a check is the more important, because many mining investors are tempted to overlook the essential proposition that a mine is "a candle, burning at both ends;" that its value is constantly diminished by its product; and that its annual profits must cover, not only a satisfactory income from the investment which has been made in it, but also the progressive repayment of the investment itself. All this sounds very elementary; yet it is too often overlooked in the enthusiasm of speculation. Each investor, if he thinks of it at all, dismisses it from his mind with the reflection that he will have abundant opportunity to "unload" during the period of dazzling prosperity which he foresees for the mine. This is the chief reason why mining has not yet become universally a regular business. That desirable result will be greatly promoted when investors purchase a mine, either with the positive intention, not of selling it out, but of working it out, or else, at a price based upon the hypothesis of such an intention. Consequently, the more this consideration is emphasized by theoretical calculations of value like the preceding, the better.

3. I should add my recognition of the elaborate paper of the late H. D. Hoskold,[2] Inspector General of Mines of the Argentine Republic, read before the Institute in October, 1902, which, by its more thorough mathematical treatment of this subject, might be held to supersede the present paper, but for two circumstances—namely, it seems to be mainly applicable to the valuation of collieries; and, by its retention of the formulas of compound interest for the calculation of sinking-funds and their corresponding annuities, renders the required calculations more complex and difficult than, in cases of less definite data, I think they really need to be. In the present paper, by abandoning the hypothesis of compound interest, I have reached a formula, precise enough for the nature of the data, and, because easier to apply, more likely in practice to be applied, than the more accurate one presented by this distinguished authority. As to the cardinal importance of a provision for the amortization of the invested capital, Mr. Hoskold sounds no uncertain note; and I may well be content to echo him.

[2] The Valuation of Mines of Definite Average Income, *Trans.*, xxxiii., 777 to 789 (1903).

The Physical Features and Mining Industry of Peru.

BY GEORGE I. ADAMS, WASHINGTON, D. C.

(New York Meeting, February, 1908.)

Physical and Climatic Divisions.

Peru is divided into three regions—the coast, the sierra, or high mountainous region, and the "*montaña*," or forest region of the Amazon basin.

The Coast.—The coast is an arid region. No rain falls excepting in the extreme northern part, which borders the Gulf of Guayaquil. There is, however, a slight precipitation in the form of mist (*garua*), which occurs on the coast hills during the so-called winter months. The aridity of the coast is due principally to the antarctic, or Humboldt, ocean-current, which, being cold and having a temperature lower than that of the land, absorbs rather than produces vapors. The coast mists occur during the season when the land is relatively cold. Near the Gulf of Guayaquil the antarctic current diverges from the coast, so that in the Tumbes valley and adjacent country the conditions in the summer months are such as produce rains. In the remaining valleys of the coast, agriculture is made possible only by means of irrigation, and is accordingly very limited in extent, although rich and varied. In the northern and south-central portions of the coast, plains extend inland from the sea, while in the southern part there is a coastal plain separated from the Pacific by a range of coast hills. Between the northern and south-central plains, as well as between the south-central and southern, the foothills of the sierra descend to the seacoast.

The Sierra.—The sierra embraces the high land dominated by the lofty cordilleras of the Andes. In the southern part of Peru there are two cordilleras, the western and the eastern, which enter from Chile and Bolivia respectively, and inclose between them the Titicaca Lake basin. To the north of Lake Titicaca these cordilleras unite in the knot of Vilcanota, south of Cuzco. From this knot three cordilleras, known as the western, the central, and the eastern, extend to the northward, reuniting in the

knot of Cerro de Pasco in the central part of the sierra. From Cerro de Pasco three cordilleras diverge. The western extends into Ecuador. The central, after it is cut at the falls of Manseriche by the Marañon, which is the head of the Amazon, dies out in the plains to the north. The eastern, after it is cut by the Huallaga at the Falls of Aguirre, unites with the central cordillera south of the Marañon. The relations of these cordilleras are shown in Fig. 1. In the southern part of Peru, where the trend of the mountains changes, producing an irregular curve, there is a high plateau to the west of the western cordillera on which a chain of volcanic peaks is found, bearing about the same relation to the cordillera as a string does to a bow. The western cordillera, in its northern part, divides for a short distance into two parallel ranges, which re-unite. The eastern is the main one, known as the White Cordillera, because of its snow-peaks, and the western is called the Black Cordillera.

The western cordillera, which is the continental divide, is a lofty range. In a list of 42 of its passes, one found in the northern part, near Huamachuco, has an elevation of about 6,800 ft. This is the lowest pass, and it will some day be the railway-route from Payta to the Amazon. The highest pass has an elevation of about 15,800 ft., and 27 other passes exceed 12,000 ft. in elevation.

The sierra, especially in its western part, is semi-arid, due to the fact that the moisture brought by the trade-winds over the Amazon valley is nearly all precipitated before crossing the cordilleras, and, moreover, the excessive evaporation due to the altitude causes the moisture of the ground to disappear rapidly. In the summer season there is a moderate rain-fall on the Pacific slope of the western cordillera. This moisture is generally considered to have come over the range from the east. The western limit of these rains is a fairly definite line, seemingly dependent upon elevation and temperature, and corresponds with the limit between the coast and the sierra excepting near the Gulf of Guayaquil, as has been noted above. In the higher parts of the sierra the temperature is too low to permit of agriculture, but products such as barley, potatoes, etc., are grown at moderate elevations and in the valleys of the inter-Andean streams, especially as they flow at lower altitudes in approaching the forest region; the agriculture, although confined to narrow limits, is rich and varied.

The Montaña.—The forest region, or *montaña*, as it is called, occupies the Amazon drainage-basin up to the timber-line.

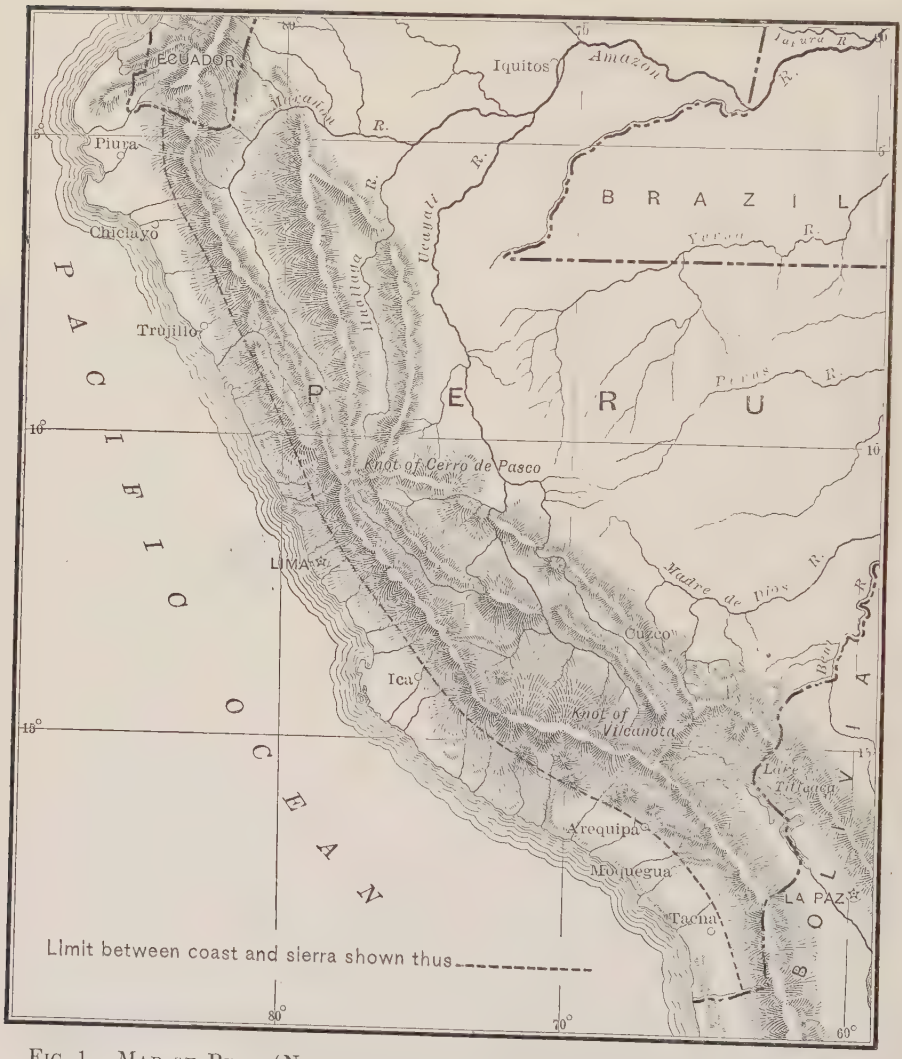


FIG. 1.—MAP OF PERU (NORTHERN AND EASTERN PORTIONS OMITTED), SHOWING THE CORDILLERAS OF THE ANDES.

There are some grassy plains, but practically the whole is a tropical forest, not yet dominated by man. The main routes of travel are the waterways, and the commerce is largely related to the rubber industry. Iquitos, on the Amazon, is the head

of navigation for ocean-steamers, but river-steamers ply much higher on the principal tributaries.

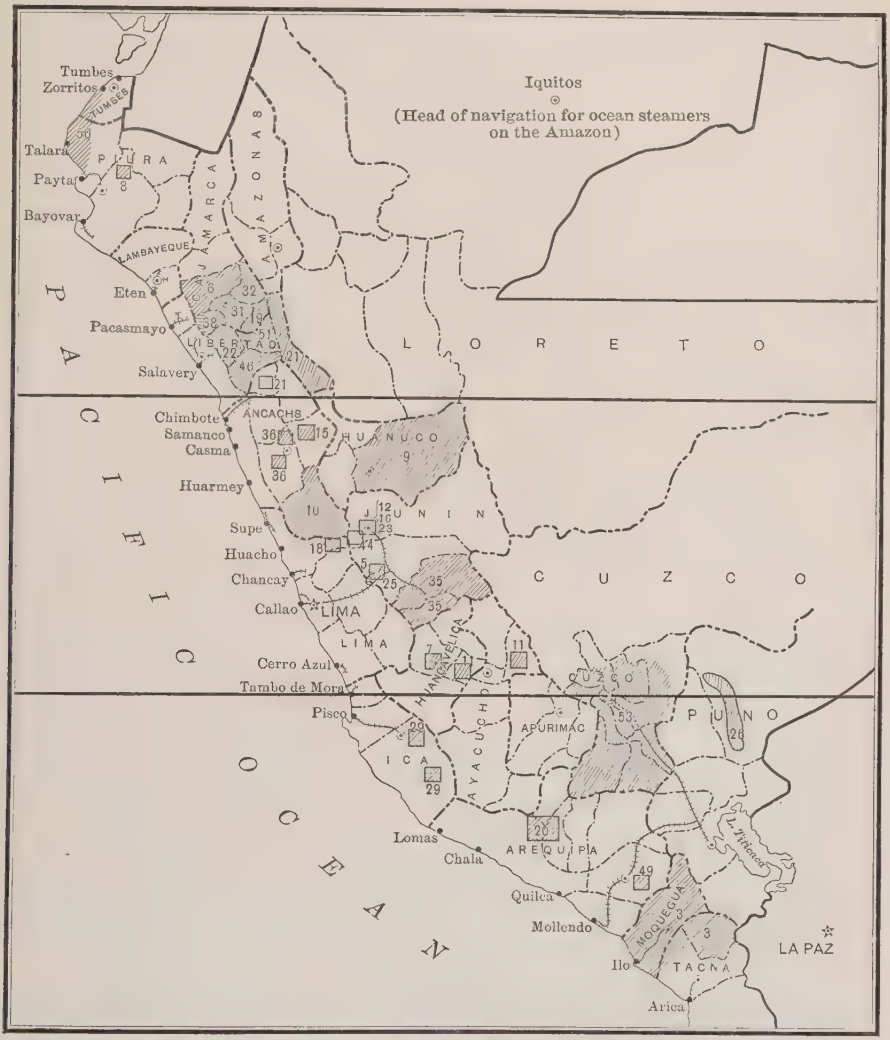


FIG. 2.—INDEX-MAP OF MINING DISTRICTS DESCRIBED IN BULLETINS OF THE CORPS OF ENGINEERS OF MINES OF PERU.

COMMERCIAL FEATURES.

Ports.—The principal port of the Peruvian coast is Callao, which is well protected by San Lorenzo Island. There are docks at Callao, which is the only port where cargo can be dis-

charged directly from the steamers, but, inasmuch as the docks are too limited for the commerce, lighters also are used. Piura in the north, Chimbote in the north-central, and Ilo in the southern portions of the coast have well-protected harbors, but, on account of the short piers, lighters are used to carry the cargoes. Many of the remaining ports have long steel piers reaching out to deep water, to enable lighters to come alongside beyond the limit where the waves usually break with much force. Mollendo is an unprotected port, with wharf piers in a very small basin. In rough weather the swells coming in from the ocean sometimes make loading and unloading from lighters in the unprotected ports impossible. This condition is a serious drawback to commerce. Moreover, the dropping of cargo into lighters requires good packing, and secure and heavy boxing and crating in order to prevent breakage.

Railways.—With the exception of the central and southern lines the railways of Peru serve the agricultural rather than the mining industry, since they do not extend beyond the valleys and plains.

The central railway, which crosses the western cordillera at an elevation of 15,645 ft., extends from Callao to Oroya, from which place a branch runs south to Huancayo. From Oroya the Cerro de Pasco railway extends northward to Cerro de Pasco.

The southern railway from Mollendo crosses the western cordillera at an elevation of 14,660 ft., and terminates at Lake Titicaca, but from Juliaca, near the lake, a branch extends to the north, reaching nearly to Cuzco, to which place it is being extended. The commerce on Lake Titicaca, which is principally with Bolivia, is carried on by small steamers. A coast railway is now being extended from Chimbote into the rich mineral districts of the department of Ancachs. The government is stimulating railroad-building by granting subsidies and concessions. The most important of these projects is a railway to connect with the central in the sierra near Cerro de Pasco, and to extend to the navigable waters of the Ucayali, a tributary of the Amazon. Provision is also made for building an extension of this line to the coast to the north of the central railway. In the south of Peru the railway from Ilo to Moquegua is to be rebuilt, and a branch from the southern railway to Ilo is contemplated in order to provide a good port as a ter-

minus for that line. The projected prolongation of the road now running from Payta to Piura will give Peru a northern railway, which will connect with the head of navigation on the Amazon.

The extension of the coast railways in general into the sierra, now in contemplation, is essential to the development of the mineral resources. In the sierra the routes of travel are usually little better than mountain trails, and the difficulty of taking in machinery and bringing out metals by pack-trains is the principal drawback to the mining industry.

The Mining Districts.—Nearly all of the important mining localities are in the sierra. The conspicuous exceptions are the petroleum-deposits and sulphur-mines in the northern coastal plains, and the gold-bearing quartz and placer-gravel deposits in the border of the forest region in the southern part of the country. The high altitude affects those not accustomed to the sierra, and it is usual for skilled and technical employees to take vacation trips to the coast.

Official Lists of Mining Claims.—The *Padron de Minas*, published May 31 and Nov. 30 of each year, contains a list of mines with a statement of the annual taxes paid or due. In the *Padron* published at the end of 1906, there were 9,789 claims listed, of which, according to the statement in the *Mineral Statistics* for that year, about 1,200 were being exploited by 160 companies.

TABLE I.—*Mineral Production of Peru for the Year* 1906.[1]

Coal,	Metric tons,	79,969	*Libras*,	138,155
Petroleum, crude,		70,832		242,542
Gold,	kg.,	1,247		170,355
Silver,	Metric tons,	230		972,958
Copper,	Metric tons,	13,474		996,055
Lead,	Metric tons,	2,568		35,125
Bismuth,				
Nickel,				
Mercury,	kg.,	2,304		495
Salt,	Metric tons,	20,226		20,226
Borax,	Metric tons,	2,598		23,392
Sulphur,	Metric tons,	1,830		2,745
Antimony,	Metric tons,	91		8,526

NOTE.—The Peruvian gold *libra* is made by law to maintain a parity with the English pound sterling. Ten silver *soles* equal one *libra*, regardless of any fluctuation in the price of silver bullion. One hundred *centavos* equal one *sol*.

[1] *Bulletin* No. 54, *Cuerpo de Ingenieros de Minas*, by German Klinge.

Mineral Production.—The mineral production during the year 1906 is shown in Table I. The coal was produced and used chiefly in the Cerro de Pasco district. The petroleum was produced at Negritos, Zorritos, and Lobitos, the refineries being at the first two places named. The principal production of gold was from the mines of the Inca Mining Co., at Santa Domingo, in southern Peru, within the border of the forest region north of Lake Titicaca. The silver production was principally from the Departments of Junin, Lima, Huancavelica, Ancachs, and Cajamarca, in order of importance. About half of the copper was produced from Cerro de Pasco and one-fifth from Yauli (Morococha), in the Department of Junin, and most of the lead was from the same districts.

Metallurgical Plants.—According to the *Mineral Statistics* for the year 1906, the following establishments were in operation during that year:

Smelting and bessemerizing of copper,	2
Smelters producing matte,	13
Lead-smelters,	3
Lixiviation,	24
Amalgamation,	17
Cyaniding,	2
Concentrating-plants not connected with smelters,	3
Petroleum refineries,	2
Sulphur refinery,	1

Of the smelting-plants, the more important are the smelters of the Cerro de Pasco Mining Co., having a capacity of 500 tons per day, situated near Cerro de Pasco; the Backus and Johnson smelter at Casapalca, having a capacity of 150 tons per day; the Huaracaca smelter, near Cerro de Pasco, having (with concentration and amalgamation) a capacity of 100 tons per day. These, together with some smaller plants, are on the central railway of Peru. Outside of this group the small smelters which were in operation are situated away from railroad facilities. The construction of some small smelters was begun during the year.

Labor.—The labor in the mines is performed by Indians and the mixed native race. They are patient and docile workmen, their principal defects being their lack of education in anything excepting Spanish methods, and their custom of celebrating

feast-days. Many of them have small patches of land which they cultivate, and, as a result, cannot be relied upon for constant work. In some localities the Indians do not speak Spanish and must be directed by men who know the Indian languages. Usually, where mining operations are carried on constantly, and the rate of pay is good, the Indians become accustomed to depend upon the mines for their living, and many of them become capable workmen. In the work connected with mining the daily pay of a common laborer ranges from 20 *centavos* to one *sol*, while in the smelting- and concentration-plants it ranges from 60 *centavos* to 1 *sol* 20 *centavos*. Packing ore costs from 40 to 50 *centavos* for a ton per kilometer. These figures are from the *Mineral Statistics of Peru* for the year 1906.

Maps.—The mother map of Peru is the one made by Raimondi on a scale of $\frac{1}{500000}$, and published in sheets for convenience in handling. The sheets are on sale in Lima at reasonable prices, excepting a few which are rare or out of print. These maps contain many inaccuracies, but are the best and, in fact, the only ones which can be obtained, excepting those published in the *Bulletins* of the Corps of Mining Engineers.

Literature.—There are many scattered references to the mines of Peru, but most of them are out of date, and many are not obtainable. The *Bulletins* of the Corps of Engineers of Mines of Peru, which have issued since 1903, are the most readily available, being published for free distribution. They are in Spanish, and not always concise or clearly summarized. For the most part they are in the form of reconnaissance reports. Some of them have been written by inexperienced young graduates from the School of Mines at Lima, but they contain important data, and usually are accompanied by small maps which show the routes to the districts.

In order to give an idea of the portions of the country covered by these reports, Fig. 2 shows the positions of the areas discussed in the *Bulletins*. In dividing the general map of the country into three parts, consideration was given to the fact that these divisions will always be more or less distinct, because of the physical features, ports, and routes of travel. Most of the forest region has no mining developments, and only a small part of it is included in the map.

The following list of *Bulletins*, each of which is supplemented

with a few explanatory notes, comprises only those of the series which relate to the mining industry.

Bibliography.

Bulletins Pertaining to the Districts on the Northern Part of the Index-Map, Fig. 2.

Bulletin No. 6. Mineral Resources of the Province of Hualgayoc, by F. Malaga Santolalla, 1904. Production of 40,000 marcos of silver on an average for each of the preceding 13 years. Lixiviation introduced in 1890. Nine lixiviation-plants enumerated. Also copper, lead, gold, and workable anthracite coal.

Bulletin No. 8. Iron-Deposits of Tambo Grande, by Pedro C. Venturo, 1904. Not developed ; no coal near.

Bulletin No. 19. The Province of Cajabamba and Its Mineral Districts, by F. Malaga Santolalla, 1905. Copper, gold, silver, lead. One lixiviation-plant and one smelter.

Bulletin No. 21. Mineral Resources and National Importance of the Province of Pataz, by Felipe de Lucio, 1905. Gold-mines and placers ; small production. Annex : Coal-deposits of Ancos in Province of Pallasca. Anthracite exploitable.

Bulletin No. 22. The Province of Otuzco and Its Mineral Districts, by F. Malaga Santolalla, 1905. Copper, silver, lead, with gold values ; small production ; one lixiviation-plant.

Bulletin No. 31. Importance of the Province of Cajamarca from a Mining Standpoint, by F. Malaga Santolalla, 1905. Copper, lead, silver, gold, antimony, sulphur, and coal. Concentration and lixiviation. Important deposits ; small production.

Bulletin No. 32. Metalliferous and Coal-Deposits of the Province of Celendin, by F. Malaga Santolalla, 1905. Copper, silver, lead, gold, and coal.

Bulletin No. 38. The Province of Contumaza and Its Mineral Districts, by F. Malaga Santolalla, 1906. Copper, lead, zinc, antimony, and coal. At present of little promise, excepting possibly the coal.

Bulletin No. 46. Mineral Resources of the Province of Santiago de Chuco, by F. Malaga Santolalla, 1906. Silver, copper, and coal. Eight lixiviation-plants ; moderate activity in mining. A promising district. Need of establishing smelting-operations.

Bulletin No. 50. Preliminary Report on the Petroleum Field of Northern Peru, by V. M. Masters ; in press.

Bulletin No. 51. Report on Mining Industry of the Province of Huamachuco, by F. Malaga Santolalla, 1907. Copper, silver, lead, antimony, and coal. One lixiviation-plant ; small production.

Bulletins Pertaining to the Districts on the Central Part of the Index-Map, Fig. 2.

Bulletin No. 5. Report on Work Done (by Commission) in the Mineral District of Yauli (Morococha), by M. G. Masias, 1903. Enumerates silver- and copper-mines, of which nine are in exploitation.

Bulletin No. 9. Mineral Resources of the Province of Huanuco, by Nicanor G. Ochoa, 1904. No producing mines ; formerly small exploitation.

Bulletin No. 10. The Mineral District of Cajatambo, by F. Malaga Santolalla, 1904. Gold, copper, lead, silver, manganese, and workable coal. Now, three lixiviation-plants, and one small smelter is being built.

Bulletin No. 12. Geological Observations from Lima to Chanchamayo, by Gustav Steinmann, 1904. General mining geology of Cerro de Pasco district.

Bulletin No. 15. Mineral Resources of the Districts of Chacas and San Luis, by Enrique I. Dueñas, 1904. Silver, lead-production; coal-deposits. Now, one smelter and two lixiviation-plants.

Bulletin No. 16. Report of the Commission of Cerro de Pasco, 1904. "Smelters in Cerro de Pasco," by Carlos E. Velarde. Nine reverberatories and one water-jacket. Cerro de Pasco Mining Company's smelter in construction, now producing. Production in year, 4,070 metric tons of copper and 7,648 kg. of silver.

"Inversion of Capital in Cerro de Pasco," by M. A. Denegri. A short history of the district and a discussion of the mineral deposits and coal-fields. Also published in *Mining World* (Chicago), vol. xxi., No. 4, p. 77 (July 23, 1904).

Bulletin No. 18. Coal-Deposits of the District of Checras, by E. A. V. de Habich, 1904. Good quality of workable anthracite.

Bulletin No. 25. Present Status of the Mining Industry in Morococha, by M. G. Masias, 1905. Copper, lead, silver. Three smelters and three concentrating-plants. One of the principal mining centers of Peru.

Bulletin No. 35. Mineral Resources of the Provinces of Jauja and Huancayo, by Enrique I. Dueñas, 1906. Copper, silver, lead, molybdenum, coal, and asphaltite; small production. Developing because of extension of railway to Huancayo.

Bulletin No. 36. Iron-Deposits of Aija and Calleycancha, by Luis Pflücker, 1906. Not exploited. Also references and data concerning mines in same region.

Bulletin No. 44. The Mineral District of Huancavelica (or Huaillay) in the Province of Cerro de Pasco, by Carlos E. Velarde. 1906. Copper and silver, coal near by. One small smelter and another building.

Bulletins Pertaining to the Districts on the Southern Part of the Index-Map, Fig. 2.

Bulletin No. 3. Report on the Coast Provinces of Moquegua and the Department of Tacna, by F. Alayza y Paz Soldan, 1903. Copper-mines and rock-salt in coast range near Ilo; copper, lead, silver, and coal in sierra; also sulphur and borax; very small production.

Bulletin No. 7. Quicksilver of Huancavelica, by A. F. Umlauff, 1904. Extensive old mines now abandoned would form basis of new industry.

Bulletin No. 11. Deposits of Tungsten at Lircay and Nickel at Rapi, by E. A. V. de Habich, 1904. Have been exploited on small scale.

Bulletin No. 20. The Gold-Deposits of Condesuyos and Camana, by L. U. Alvarado, 1905. Quartz-veins exploited on a small scale.

Bulletin No. 26. Report on the Auriferous Deposits of (the province of) Sandia, by Luis Pflücker, 1905. Reconnaissance of the District of Poto (ground-sluicing and monitors), Aporoma, Cotani, Montebello, and Chailluma (ground-sluicing); small production.

Bulletin No. 49. Borax-Deposits of Laguna de Salinas, by Alberto Jochamowitz, 1907. Moderate exploitation.

Bulletin No. 53. Mining Aspect in the Department of Cuzco, by Enrique I. Dueñas, 1907. Copper-, silver-, lead-, and gold-prospects; also auriferous gravels; no production. Refers to newly-discovered petroleum field at Pusi, on border of Lake Titicaca.

The Mineral Resources of Korea.

BY HALLET R. ROBBINS, A.B., B.Sc., LONDON, ENG.

(New York Meeting, February, 1908.)

KOREA, the ancient "Hermit Kingdom," is a peninsula jutting out from the coast of eastern Asia. By the natives it is called "Chosen," which, translated, means "Land of the Morning Calm." It lies between the Yellow sea and the Sea of Japan. On the north it is separated from Manchuria and the Russian province of Primorsk by the Yalu and Tiumen rivers; on the south the Korean straits interpose a rough voyage of 120 miles to the nearest point in Japan. The entire country is situated between the parallels of 34° and 42° N. latitude.

The surface of the country is made up of successive ranges of unglaciated mountains, separated by narrow, fertile, and more or less densely populated, valleys.

Through all the obscure centuries of its history, Korea has succeeded in maintaining a monastic seclusion, over which, from time immemorial until the late Chino-Japanese war in 1894, China exercised a shadowy sovereignty.

Although Korea had been forced into slight foreign contact previously, her first foreign treaty was signed with Japan in 1876.

Under the restraining guardianship of China, the Japanese treaty bore but insignificant innovations for Korea; but when, in 1883, a treaty was made with the United States, the first of the western nations, a decade after the American gunboats, under Commodore Rodgers and Lieutenant Schley, forced their way up the Salee river, Korea awoke from her sleep of 4,000 years.

The Korean peninsula suddenly became the arena where the enterprises and individuals of many foreign nations met in fiercest rivalry. Out of this struggle grew two singularly successful American enterprises, both of which have played an important part in developing the natural resources of the country.

The first, and best known of these, is the Oriental Consolidated Mining Co., which was organized by Leigh Hunt

(well known in connection with cotton and irrigation enterprises in the Soudan), with the co-operation of W. D. Townsend, of Chemulpo. Mr. Townsend was one of the first white men to travel in the interior of Korea, and the very first to visit the mines in the Unsan district. Dr. H. N. Allen, then American minister, being on very friendly terms with the Emperor, secured for these Americans a concession for all mineral and timber lands in that district, which is about a day's journey NNE. of the town of Anju, and about 100 miles north of the city of Pingyang. The area of the concession is not accurately known, never having been surveyed; but it is approximately 275 sq. miles. Within this area are 10 separate gold-mines, of different sizes and richness, as well as some deposits of very low-grade auriferous gravel. The work of this company began at the Chittabalbie mine with a 10-stamp mill, operating on $50 rock. This mine is now completely exhausted, after producing $1,500,000 worth of bullion. Other mines have been and are now being opened from time to time in different parts of the concession; and there are to-day 5 or 6 producing mines, two of which, Taracol and Tabowie, are equipped with 80-stamp mills. At Taracol there is also a cyanide-plant for treating the concentrates from the mill. This camp, the largest on the concession, contains blacksmith-, machine-, and carpenter-shops, store-houses, a bunk-house, "chow-house," etc. A view of the camp is given in Fig. 1.

The total amount of bullion produced to date is about $9,500,000; and the net profit to the company has been considerably over $250,000 per year, which represents a very large percentage on the capital actually invested in plant and equipment. The value of the ore now worked averages about $5 per ton. At the Taracol mine it is about one-half free-milling, and about 81 per cent. of the values in the remainder is extracted by leaching with potassium cyanide in the ordinary manner. The labor question has afforded some perplexities in the past, but, on the whole, has been comparatively simple. Koreans make excellent underground miners, and are used exclusively for this purpose. All mining is done by hand. For ordinary unskilled labor on the surface, and for certain kinds of skilled labor, the Chinese have proved themselves greatly superior to the Koreans. These Chinese come largely

from Shantung, the province from which most of the Rand contract-coolies were recruited. They are very industrious and easy to handle. The rate of wages for Korean miners and Chinese coolies is about 23 cents per day of 10 hr. As to skilled labor, Japanese carpenters receive about $1.50 per day. Chinese machinists, engineers, bricklayers and stone-cutters are obtained at from 35 cents to $1 per day. Ore is mined for from 75 cents to $1.50 per ton, depending on local conditions at the mines. The cost of milling is about 58 cents per ton, nearly half of which is for fuel. The cost of cyaniding at the Taracol plant slightly exceeds $2 per ton of mixed concentrates and heavy sands. These low costs appear to be due largely to the cheapness and efficiency of the native and Chinese labor, and to the competency and faithfulness of the white shift-bosses and foremen, rather than to the somewhat inconsistent and over-conservative management.

The main sources of loss are inadequate protection against fire, inefficient metallurgical treatment, and the numerous changes in the corps of white employees, of whom there are from 60 to 75. A large dam was constructed to impound water for power; but, owing to some flaw in the design, an insufficient wasteway was provided, and just as the dam was nearing completion, the seasonal rains came on and washed it away. Its replacement was a heavy burden of expense, and the method of construction, that of hand-packed rock-fill with a timber apron, does not seem to have been the most economical for this location, where conditions were favorable to a loose rock-fill dam with a concrete core, or even to an earth dam with core or puddled wall.

However, the enterprise has proved very profitable since its inception, although the future prospects are hardly as encouraging as might be wished. Coal is not available for fuel, and the company is obliged to depend on local woodland for cordwood and mining-timbers. Practically all the trees have been cut from the concession of the company, at least in the vicinity of the mines.

The other great American enterprise in Korea is that of Collbran & Bostwick. Originally railway and general contractors, this firm has extended the sphere of its activities over a wide field. It built the first railroad in Korea, the

Seoul-Chemulpo line, 26 miles long, which crosses the Han river by a steel through-truss bridge of 9 spans. This line was well constructed and of standard gauge; and so well did it please the Japanese that they bought it, and built the rest of the extensive Korean railroad-system to match it. One may to-day ride through the entire length of Korea over as good a roadbed, and in cars as comfortable, as on any standard American railroad, while in Japan the trains are narrow gauge and built after the English pattern, so that they impress the traveler more as toys than anything else. A view of Chemulpo harbor is given in Fig. 2.

The same firm built and now operates an electric street-railway in Seoul, which gives the 250,000 inhabitants excellent service to all parts of the city and suburbs at a very moderate fare. An interesting photograph of one of these cars is given in Fig. 3. The cars are well patronized by the natives, and the system yields a large revenue. Electric light and power are supplied to private consumers about the city, including the residences of foreigners, foreign hotels, foreign consulates, Korean government buildings, etc. Moreover, the construction of a complete water-works system for the city of Seoul has just been completed by the firm. This system will supply all quarters of the city with an abundance of pure water. The water-supply of the city has, heretofore, been derived from shallow wells, which were all contaminated by seepage from the neighboring open sewers; and epidemics of water-borne diseases have been of annual occurrence. The water was conveyed from the wells to the houses of consumers by carriers, each bearing two wooden buckets or coal-oil tins suspended from a framework on his back. No sanitary precautions were observed in handling the water, and every opportunity for contamination from dust, etc., was afforded.

Collbran & Bostwick conduct also a large banking business, supplying capital to native and Chinese merchants and others who prefer to deal with the American bank rather than with the Japanese bank.

By no means the least important of the firm's activities are its mining interests. It has already done some work in the development of a gold-mine at Suan, about 60 miles E. of Pingyang, in which there is now in sight a considerable body

of ore assaying from $5 to $12 per ton. Conditions here are favorable for economical operation, and plans are under consideration for the erection of a stamp-mill in the spring. The firm is also interested in a copper-mine in the north-central part of Korea, which is very favorably situated for economical operation, except for the matter of transportation—being five days' journey over a rather rough trail from the coast at Puk-Chun. It is, however, in the center of the only extensive forest-area in Korea; and an abundant supply of charcoal and mining-timber is thus assured. Considerable work has been done here by native Koreans, who mine the ore with hammer and gad, roasting it in crude stalls built of stones. The smelting-operation is somewhat interesting. Charcoal only is used as fuel, and the blast is produced by four coolies, two on each end of a sort of piston-rod which projects from each end of a rectangular wooden box, about 18 in. wide, 3 ft. high, and 5 ft. long, with flap-valves made of stiff paper or cardboard in each end. A wooden piston moves back and forth in the box as the men push and pull on the rod, and a blast of surprising pressure and steadiness is produced. The smelting-furnaces are difficult to describe. In general, they resemble a blacksmith's forge.

The work of the natives has developed a promising body of pyritic ore, assaying about 12 per cent. of copper, and carrying appreciable values of gold and silver. Water-power may easily be secured in the neighborhood, since the rainfall is abundant, and the streams have a rapid fall toward the Yalu river. Development-work at this mine has just been started by Collbran & Bostwick's engineers, but on a small scale, owing to the activity of Korean revolutionaries in this section.

This copper-mine is in the center of the tiger district. It is not generally known that real "life-sized" tigers inhabit so cold a climate as that of Korea and Manchuria, but it is a fact that tigers as large and ferocious as those of the Bengalese jungles are found in these regions. While I was stopping at this copper-mine, I was awakened, in the dead of night, by a tremendous commotion outside my door. Natives were shouting, Japanese soldiers were firing their rifles and revolvers, and a crimson glow lighted up the surroundings. Asking from my interpreter the cause of the excitement, I

was informed that five tigers had been observed prowling about the outskirts of the village, and that the noise and fire had been created with the view of scaring them away. Next morning we learned that the fresh-picked bones of three Korean wayfarers had been found not far away. I have seen tiger-tracks measuring about 5 in. in diameter; and I always carried my Smith & Wesson "44" ready for prompt use if required. Tiger-hunting is a favorite sport among the more adventurous Koreans, as a tiger-skin will bring from $50 to $100, upon which a native may live for a year in idleness and luxury.

Fig. 4 is a view of the mining-camp at Kojimdong, and Fig. 5 shows the crude method of roasting the ore in stalls.

Several other foreign mining enterprises, established in Korea, have met with varying degrees of success or failure. An English company secured a concession for a very promising gold-mine at Gwendoline, near Pingyang, but were deceived by the erratic form of the ore-body into investing a very large sum of money in plant and underground prospecting. The deposit suddenly pinched out at a comparatively shallow depth, with the result that the company suffered a net loss of nearly 1,000,000 yen ($500,000).

A German firm secured a concession for some placer-ground in the province of Kang-won, but its operations were conducted at a considerable loss. Report has it that crafty natives salted the samples taken by the company's engineers before the concession was secured.

An Italian syndicate has a concession for some gold-prospects near Sen-chun in the province of Pyong-an, but, at last accounts, active work had not begun. At one of these prospects I saw a very clever piece of native workmanship. It was a small stamp-mill, driven by an overshot water-wheel, complete in every respect, and made entirely of wood, with the exception of the shoes and dies. Three stamps were dropping and crushing ore in a slow but effectual manner. Mercury was not used to amalgamate the free gold; but the pulp as it came from the mortar was panned by hand. No statistics were available as to the duty of the mill, or the assay-value of the heads and tails; but it did not seem probable that a large profit could be realized from the operation.

Another foreign enterprise worthy of mention is at Chiksan, in the province of Chung-Chung. Here a gold-bearing formation may be traced for a distance of several miles along its strike, and development is now being carried on with joint American and Japanese capital, under the effective management of A. W. Taylor and E. W. Mills, Americans of long experience in Oriental mining and metallurgical work.

The native methods of mining are very crude, and only a small proportion of the values is recovered. The Koreans do not use powder in mining. The ordinary method of procedure in the case of auriferous quartz is to build fires against the face of the ore-body, then throw cold water on the heated rock, and break off as much of it as possible with hammer and gad. The ore is then ground to powder between two flat stones, something after the manner of "bucking down" assay-samples, and this powder is panned for its free gold. Of course, none of the gold associated with the sulphurets is recovered; and, since about one-half the gold in many Korean ores occurs in this condition, the wastefulness of the process is apparent. The development operations of the natives are frequently very laborious. At one place I saw a tunnel about 300 ft. long driven through granite to cut a copper-vein in depth. The only tools used were the hammer and gad, and no fires were employed to heat and disintegrate the rock. The tunnel was of standard cross-section and very neatly trimmed to a smooth surface. Unfortunately, the copper-vein, at the level where this tunnel cut it, proved too low in grade to be workable; so the project was a dead loss to the Koreans who financed it.

Placer-gold is found in almost every prefecture in Korea; but practically all the deposits are of too low a grade to attract foreign capital, and the richer ones have long since been worked out. Gold-dust was demanded by China as her tribute from Korea during the long centuries of the latter's vassalage to the Sons of Heaven.

In the native method of placer-mining, a small stream of water is conducted through a ditch about 1.5 ft. wide and 3 ft. deep, which is lined with stones. The auriferous gravel is excavated with small picks (the boulders being removed by hand), and shoveled into the ditch. The gold is recovered by panning the

FIG. 1.—TARACOL CAMP, ORIENTAL CONSOLIDATED MINING CO., UNSAN DISTRICT, KOREA.

FIG. 2.—CHEMULPO HARBOR, KOREA.

FIG. 3.—AMERICAN ELECTRIC CAR IN SEOUL.

FIG. 4.—KOJIMDONG MINING-CAMP.

FIG. 5.—NATIVE FURNACES, KOJIMDONG.

FIG. 6.—PLACER-MINING OPERATIONS IN KOREA.

FIG. 7.—PLACER-MINING OPERATIONS IN KOREA.

FIG. 8.—A GROUP OF MAFOOS.

concentrates removed from the stone sluice, in large wooden pans. No mercury is used for amalgamating the gold. The native panners seem very careless; and a large part of the very fine gold is lost. Figs. 6 and 7 illustrate the general method of placer-mining in Korea.

Native metallurgical operations have gone into decadence in modern times, but there are abundant evidences that the ancient Koreans were very skillful in the art. For example, near the town of Kyong-ju there hangs to-day, on its great beam, a bell as large as the great bell of Moscow, which still gives forth its summons as deep and clear as the day on which it was cast, 1,400 years ago.

Native mining operations are to-day confined to gold placer-mining in the manner described, copper-mining and smelting at the mine referred to, work upon a few small auriferous quartz-deposits, and iron-smelting on a small scale in several places. So far as I am aware, there are no producing silver-mines, though I have examined several which have been worked by the natives in the past. It may be noted in passing that nearly all the gold-quartz veins in Korea occur in a granite country-rock—an association which is somewhat unusual, though not unknown, elsewhere.

As for coal, there are some mines near Chinnampo belonging to the Korean Imperial Household, which have been worked intermittently, under foreign supervision. The poor quality of the coal, combined with alleged graft in the management, has prevented the enterprise from attaining notable success. All the coal burned on the Korean railroad-system is brought to Fusan by boats from the famous "harbor" mines of Nagasaki, Japan.

Any considerable extension of either native or foreign mining enterprise in the future is not likely. The whole country, except in the extreme north, has been thoroughly examined by native and foreign prospectors, and most of the mineral deposits have proved to be too low in grade to invite development.

In company with an Australian mining engineer, I made a four months' trip through the northern central part of the country in the summer of 1906; and although several gold-, silver- and copper-prospects were examined, not one of them was sufficiently rich to warrant us in recommending it for pur-

chase and development. A description of the organization of our party may be of interest. We traveled with a pack-train of about 12 ponies, each led by his owner, called a "*mafoo*," who was paid according to the distance traveled, at a rate which varied from 3 to 9 cents per mile, according to local conditions in the different parts of the country. Out of this amount the *mafoo* was obliged to feed and house himself and steed, and support himself on the return journey to his home. A photograph of the *mafoos* attending the party is represented in Fig. 8. An average day's journey was from 25 to 30 miles. There are few roads in the country; and practically all our traveling was done over rough mountain-trails. We had an interpreter, to whom we paid $10 per month and his traveling expenses; an excellent cook, who received $7.50 per month and expenses; and a "boy," who received the same. All of them spoke more or less English, and each was more or less proficient in the duties of the others. We two white men and our three servants rode on ponies lightly laden with bedding, etc., which, when skillfully disposed on the pack-saddle, made a very comfortable seat. The other ponies were laden with canned provisions and other necessary supplies. Some fresh provender could be purchased along the route—*e.g.*, eggs at 5 or 6 cents per dozen, and chickens at from 10 to 12 cents each. Upon arriving at a village where we intended to stop for the night, the interpreter or the cook selected the house which looked most inviting, and calmly installed our baggage in it without deigning to consult the wishes of the proprietor. If the latter offered objections (which was rarely the case), he was summarily ejected; and on our departure the next morning received about 10 cents to assuage his wounded feelings. If, on the other hand, he showed himself courteous and obliging, he was given a tip of 25 cents or half a dollar. We carried folding canvas cot-beds, with mosquito-net covers, as well as all cooking utensils, a wooden bath-tub, camp-stools, etc. Each white man had one pack-box about 1 ft. wide, 15 in. high, and 2.5 ft. long, in which to carry his belongings. All our pack-boxes were of this size, which proved a very convenient one. Bed-bugs were frequently encountered; and the first question asked of a prospective host was "Pintai isso?" which means "Bed-bugs have got?" The answer was too often "Yes;" and we became so familiar with

them that we quite missed their friendly society when we returned to the effete civilization of Seoul. In several districts we were the only white men the natives had ever seen, and our woollen clothes, food, and manner of eating were objects of the liveliest curiosity. The Koreans are the most inoffensive and good-natured people in the world, and, as far as they are concerned, one may travel anywhere in Korea with perfect security.

As laborers they are much inferior to the Chinese, except in underground mining, for which they seem to have a natural aptitude. The average Shantung coolie I consider the most efficient laborer in the world, if under proper supervision. He is strong, energetic, good-natured, naturally full of fun, and usually very tractable. The best feature of the labor-situation in Korea is the fact that Chinese and Koreans will never pull together, and a strike by one nationality may always be broken by employing the other.

To sum up the present industrial condition of Korea, it offers, on the whole (or at least it did offer, before the Japanese seizure of the country), an attractive field for carefully managed foreign enterprise. The labor question, as has been explained, is simple; the climate is very healthful, and does not differ greatly from that of New York; water is abundant, and power may be developed from it almost anywhere at small expense. Transportation is by bullock-cart or pack-ponies. Fuel and timber are scarce; and this is the most serious drawback (aside from the political situation) to the successful prosecution of mining enterprises, and presents a question into which examining engineers should look most carefully. Where timber is found, it is usually oak or pine. As remarked above, the only extensive forest-areas are in the extreme north of the country, near the Manchurian frontier.

A paper dealing with Korea would be incomplete without some reference to the present unfortunate political situation. Under this head, however, I must be content to refer the reader to such recent and impartial books as: *Tomorrow in the East*, by Douglas Story; *The Passing of Korea*, by H. B. Hulbert; *The Truce in the East*, by B. L. Putnam Weale; and *The New Far East*, by Thomas F. Millard. Time will show whether, as these writers apprehend, the Japanese ascendency will crush the enterprise of other nations, or will really furnish for it the promised "open door."

The Bogoslovsk Mining Estate.

BY WILLIAM H. SHOCKLEY, TONOPAH, NEV.

(New York Meeting, February, 1908.)

I. Introduction.

There was an extensive mining and industrial exploitation of Russia, about 20 years ago, by Belgian, French and British capitalists; but the results were discouraging. It is said that the Belgian and French public together lost approximately $400,000,000, chiefly in the coal and iron industry of southern Russia. This loss is largely accounted for by the fact that most of the companies were formed, as usual in "boom" times, to sell stock; and that thorough investigations were not made.

In 1906, renewed interest was taken in Russian mines, especially in those of gold and copper, and a considerable literature is growing on this subject; but as yet I have seen no account of one of the large Russian mining estates. The following notes on such property, where the conditions are so different from those in the mining regions of the United States, should be of interest:

This inadequate and probably somewhat inaccurate account is made up, partly from my own notes made on the ground, in the summers of 1904 and 1905, and partly from a volume on the geology and resources of the estate issued several years ago, containing surface- and mine-maps, and from a pamphlet published in 1904, giving a general account of the condition and prospects of the property. This pamphlet includes a report made in 1899 by Arthur L. Pearse, a member of this Institute. Both these books are in Russian, and were prepared by the Bogoslovsk Mining Co., which owns and operates the property.

Articles in many other languages have appeared dealing with various matters connected with the estate; but I know of no other general account of it. In numerous books and articles on the Urals and Siberia, however, localities are described in which somewhat similar conditions prevail. Recent papers by

Dr. F. H. Hatch and A. L. Simon in the *Transactions of the Institution of Mining and Metallurgy*, contain notes of special interest to mining engineers. See also my paper on Gold-Dredging in the Urals.[1]

The Bogoslovsk estate lies in the low rolling hills east of the summit of the Ural range, and is geographically in Siberia, though politically in the Perm government of European Russia. Its irregular area of 3,600 sq. miles extends 144 miles N–S., with an extreme breadth of 53 miles; the town of Bogoslovsk, near the center, being in lat. 60° N., long. 60° E. The estate also owns some outlying areas. While exceeded by the immense private properties of the Czar and by some others (such as that of the Prince of Thurn and Taxis, which is said to comprise 40,000 sq. miles, of which 17,560 belong to the Domains Co., Ltd., of London), this is one of the largest in the Russian Empire and ranks with the great estates of the world.

I do not know the exact conditions under which the property is held: 1,469 sq. miles are forest-lands, rented from the government for 100 years, with certain rights over the minerals; and the remainder seems to be owned in fee simple.

The forests are considered by local experts to constitute its chief wealth. Since the building of the branch railway connecting with the main lines at Goroblagodatskaia, several sawmills have been started, and some of the lumber will be shipped to London via Archangel. The forest is used at present mainly to supply charcoal, the chief fuel for iron- and copper-smelting.

Among the other resources are deposits of vein- and placer-gold, and of platinum; ores of copper, manganese, iron and chromium; beds of lignite; refractory furnace-materials; limestones; and cement-rock.

The company makes pig-iron from its ores in charcoal blast-furnaces, and converts this into steel, which it manufactures into rails and structural forms; smelts the copper-ores; and makes sodium and potassium bichromate, fire-brick and furnace-linings. Glass-, cement-, and phosphorus-works have been established, but are now idle.

The estate was famed for its riches in iron and copper in the last half of the 18th century, when iron- and copper-smelting

[1] *Trans.*, xxxvii., 322 to 330 (1907).

was begun by private individuals, to be continued later by the Crown, with convict-labor. A few exiles are still sent to this region. They are not allowed to cross certain boundaries, but are free to choose their occupations. In 1875 the estate passed into private hands, and in 1885 it was given by Baron Stieglitz (who had bought it for $3,000,000) as a birthday present to his daughter, who has retained the ownership, operating the property since 1895 through the Bogoslovsk Mining Co., which she controls. It is said that she has never visited the tract, though her husband was there once for a few days, and her sons have been there for longer periods; and that she has never received a kopeck of income from it—the reason for this being, that any profits realized are at once put into improvements, which constantly call for fresh expenditures. The capital already employed is about $15,000,000, and much more could be used with advantage. Fig. 1 shows the residence of the General Manager, and other buildings, and Fig. 2 is a view of the town of Bogoslovsk as seen from the smelter.

II. Management.

A notion of the many activities comprised in this administration may be gained by analyzing the general expenditures, stated for 1904 at $164,285, according to the *smieta* or estimate for that year. Such an estimate, comprised in two or more printed books, and covering in detail all the operations of the business, is prepared at the close of each year, and all the expenses of the next year are predicted to a kopeck. The remarkable thing is, that, as a rule, the results agree very closely with these predictions, the difference between the expected and the actual amount of ore produced in the mines, for example, being only a few per cent. In this particular, such a correspondence means, of course, that the mines are worked in a leisurely manner, and that the mine-managers always have ore-reserves, or stocks on hand, to cover all possible delays—something after the manner of the "secret" gold reserves of Australia and the Rand. In my judgment, this detailed preliminary estimate is probably, on the whole, a detriment to the proper working of the mines, making the engineers in charge too cautious, and retarding the development of the properties; it has, however, the advantage that the owners are not called

on unexpectedly for large sums of money to meet unexpected emergencies or new improvements suddenly called for.

The estimate of general expenses for 1905 was as follows:

Central-office expenses,	$26,204
Post and telegraph,	3,800
Messengers,	4,347
Buildings and rooms,	4,841
Fire-prevention,	1,640
Mining-school,	4,650
Children's asylum,	2,799
Old men's home,	806
Geological museum,	3,350
Technical bureau,	5,149
Library and reading-room,	1,113
Telephone service,	6,542
Repairs to buildings,	3,908
Roads and bridges,	690
Medicines,	21,068
Taxes,	31,570
Insurance,	20,000
Help to unfortunates,	8,250
Flour,	3,500
Wages to mining students,	1,580
Sundries,	8,478
	$164,285

In this statement, "fire-prevention" refers, for the most part, to forest-fires; the "mining school" covers a general as well as a technical education; the "geological museum" represents principally the geological survey of the estate and mines, and makes a specialty of petrography, having 80,000 rock-slides on hand at the end of 1904; free medical attendance and free medicines are both, I believe, required by law; and the "wages to mining students" are paid to students from the Royal School of Mines in St. Petersburg, who act, during the summer vacation, as assistants to the engineers, studying special problems and receiving $25 per month salary. They add much to social life during their visit. By reason of political troubles at their school, at the time of my visit, a number of them had been requested to remain on the estate for a year. The managers, however, did not seem desirous to have this number increased.

The Russian mining laws should be carefully studied by foreigners mining in Russia, and pains should be taken to keep on the right side of the officials. The subordinates do not

object to presents; but it is not wise to attempt to purchase immunity for violations of the law. The most alarming provision of the law is that which makes a manager personally liable for accidents, even punishing him with imprisonment. This, however, can be avoided by having a Russian nominal manager to bear this responsibility. Foreigners engaged in mining in Russia tell me that they have very little trouble with the laws.

III. General Conditions and Labor.

The climate is healthy. There are no peculiar local diseases. The maximum summer temperature is 90° F.; the lowest winter temperature, —55° F. Winter is the active season for forest-work, as the roads are almost impassable swamps in summer. Insect pests are troublesome in summer. The rivers abound in fish, and the forests in berries. Owing to lack of enterprise, there is but little agriculture on the estate; but many crops could be grown, and it is an excellent dairy country. The average rainfall for several years was 15 inches.

There is a population of 35,000 on the estate. The greater portion of the hard work is done by Tartars from the province of Kazan on the Volga, who usually remain on the estate but a few months at a time, their movements being regulated by the seasons of planting and harvest. They are excellent workers, but poorly trained. Until 1906 no strikes had been known; but in that year the general unrest reached the Urals, and there a number of (not very serious) strikes took place. As a rule, the workmen are very patient, and make no trouble, even if their wages are some months in arrears.

The usual dwellings are log houses, and those of the managers and engineers are very comfortable. At some of the mines, the workmen are housed in large buildings of one room with bunks around the sides. The single men occupy one building, and the married men with their families another. Women work on the surface but not underground in the mines; they sort ore, drive carts, and labor with the leasers on the placers.

The natives of the Urals are noted for their independence. They are much more prosperous than the agricultural laborers of Central Russia.

The following statements will show the general conditions of living:

Prices in Bogoslovsk in 1904.

Dynamite (92 per cent. gelatine),	$0.45 per lb.
Fuse,	0.35 per 100 ft.
Caps,	1.15 per 100.
Wood,	1.50 per cord.
Coke,	13.85 per 2,000 lb.
Charcoal,	4.25 per 2,000 lb.
Candles,	0.10 per lb.
Flour,	55.00 per 2,000 lb.
Butter,	$0.20 to 0.25 per lb.
Potatoes,	8.30 per 2,000 lb.
Beef,	$0.04 to 0.06 per lb.
Mutton,	none
Pork,	none
Ham,	0.14 per lb.
Vodka,	0.35 per liter.
Quass,	0.20 per pail.
Rye,	27.50 per 2,000 lb.
Partridges,	0.15 per pair.
Capercailzie,	0.40 each.
Wild geese,	0.40 per pair.
Salt, coarse,	8.30 per 2,000 lb.

Wages at the Frolovsky Copper-Mine.

Foremen,	$22.50 to $25.00 per month.
Shift-bosses,	15.00 to 17.50 per month.
Time-keepers,	9.00 to 12.50 per month.
Clerks,	50.00 to 150.00 per year.
Hoisting-engineers,	20.00 per month.
Pumpmen,	12.50 to 17.50 per month.
Wood-carriers,	10.50 per month.
Boys,	5.00 to 7.50 per month.
Blacksmiths,	12.50 per month.
Blacksmith-helpers,	10.00 per month.
Head machinists (including room with fire),	32.50 per month.
Shaft-men,	0.43 per diem.
Mine-drillers,	0.38 per diem.
Surface-drillers,	0.30 per diem.
Mine-helpers,	0.30 per diem.
Surface-helpers,	0.25 per diem.
Laborers (12 hr.),	0.35 per diem.
Laborers (ordinary),	0.25 to 0.30 per diem.
Women laborers,	0.15 per diem.
Surface-machinists,	0.33 to 0.63 per diem.
Timbermen,	0.45 per diem.

The Frolovsky mine employs 540 men in winter, with 6 clerks in the office, and 52 mechanics. I have no record of

the administrative force of the estate; but there is a small army of officials, all of whom, as well as the students, wear uniforms prescribed by law. Fig. 3 shows two mining students in the prescribed dress.

The chief Russian characteristic which I noted was an easy, good-natured way of doing business, and an absence of rush; no "strenuous" people were wanted. None of the copper-mine managers had ever visited mines outside of Russia.

Russian holidays are a great detriment to steady work. Including Sundays, there are nearly 100 in a year; and at Easter, after the long Lenten fast, it is considered a duty to drink heavily. When, in 1904, the copper-smelter was kept going during the Easter holidays, it was regarded as a great feat. I saw no drinking, however, among the officials on the estate.

Late reports (August, 1907) from the Urals indicate that there is a strong revolutionary spirit in the air, and that the workmen are giving much trouble.

IV. Transportation.

Transportation between the chief mines and works is performed by the company's 100-mile narrow-gauge railway. The Nadesda works are connected with the main-line Russian railways by a broad-gauge railway 123 miles long, completed in 1905; 8,000,000 roubles (about $4,000,000) having been advanced to the company by the Russian government. Before that time, most of the products of the estate were transported by the company's fleet of steamers, the most distant ports being Semipalatinsk on the Irtish river, and Bieska on the Ob river, each at the end of a voyage of 2,300 miles. Semipalatinsk is approximately in lat. 50° N., long. 80° E.; and Bieska (Busk) in lat. 53° N., long. 85° E.

As this instance shows, the inland water-ways of the Russian Empire, with their vast traffic, are a marvelous advantage, and will be of the utmost importance to the commercial development of the country.

The company's fleet comprises 10 steamers, with a total capacity of 21,000 tons, and 54 barges and boats with a total of 41,000 tons capacity. The freight-movement for 1902 was 117,000,000 ton-miles.

V. Forestry.

The comparatively level surface of the tract is thickly covered with larches, firs, and pines; birches and poplars are less abundant; oaks are wanting. Under the management of experienced foresters, the trees are cut from blocks of 40 acres, surrounded on all sides by uncut forest. On these soon appears a new growth, which will be ready for cutting again after from 80 to 100 years. Some of the blocks thus cleared in 1904 had been cut for the first time in 1819. Under this system, the 1,412 sq. miles of forest are capable of producing 279,000 cords of wood annually. Including 1,150 sq. miles of rented lands the total annual capacity (in other words, the total annual growth) is 549,000 cords. According to the latest data in my possession, 6,000 men employed during the winter in this country, using 6,000 horses, cut and hauled 416,000 cords, or 133,000 cords less than the full forest capacity. The charcoal, made in heaps and in modern kilns, costs $4.25 per ton of 2,000 lb.

VI. Iron and Steel.

The chief industry of the estate is iron-making in charcoal blast-furnaces and the manufacture of steel. The product in 1905 was 75,000 tons, nearly all of which was made into steel. Both the iron and the steel are of exceptionally good quality. The latter is made into rails and structural shapes at the Nadesda works. In 1903 the product was:

54,000 tons of iron, costing,	$8.10 per ton.
63,000 tons of steel, costing,	13.70 per ton.
44,000 tons of rails, costing,	21.08 per ton.

Extensive alterations and additions, since made to these works, have increased their capacity and reduced the cost of their product. It was estimated that, after completion of the works, the iron would cost $6.85 per ton (a figure which, at the present time, is probably surpassed in cheapness only by the iron-furnaces of Shansi, China, where excellent iron is made for $5 per ton), and that the annual product would be 83,000 tons, or one-sixth of the total product of the Urals. Fig. 4 is a view of the iron blast-furnace at the Nadesda works.

In 1905, the motive-power employed at Nadesda amounted to 4,555 h.p. But a gas-engine of 6,000 h.p., made by the

Cockerill Works of Belgium, was then being erected, to replace the steam-engines, and to be run by the waste gases from the iron blast-furnaces. The power in excess of that employed in the iron industry was to be electrically transmitted to the copper-mines. The erection of this large gas-engine, of most modern type, is a creditable illustration of the enterprise of Russian managers in this region. Other instances could be given. Steam-engines were used in the Urals in 1766; and some of the earliest (as the Russians claim, the very earliest) experiments in copper-converting were made there. The Bogoslovsk copper-converter was set up in 1882.

The charcoal pig-iron made on the estate is converted by the open-hearth process into steel of excellent quality, which is, for the most part, rolled into rails and structural shapes.

The quantity of iron-ores developed, including those of the the Northern iron-mine on the Lozva river, has been estimated by Professors Feodoroff and Nikitin at 8,000,000 tons. These ores are hematites and limonites, containing from 30 to 66 per cent. of iron, and occurring in basin-deposits, from 13 to 65 ft. thick, from 200 to 650 ft. wide, and from 500 to 1,600 ft. long. There is believed to be a great deal of undeveloped ore on the estate and in the Northern iron-mine. The output of ore from 1896 to 1903 was 686,000 tons, mined at a cost of from $1.15 per ton in 1896 to $0.81 in 1902.

The deposits are worked by pick and shovel, the ore being hauled from the quarries in 1,000-lb. loads by wagons, often driven by women, as shown in Fig. 5.

VII. Gold and Platinum.

Gold is found in both placers and veins. From 1895 to 1902, the product of the estate was $1,750,000. The placers all carry platinum, of which 2,517 oz. Troy was produced during the period named.

The productive Voronsovsky iron-mine was discovered in searching for a gold-bearing quartz-vein. In operating the iron-mine, this vein has been exposed for a horizontal distance of 700 ft., with an average width of 4 ft. It contains a soft clayey material, carrying "bunches" of "rotten quartz," the treatment of 601 tons of which is reported to have yielded $5.87 per ton.

The Alexandrovsky quartz-mine, near the Voronsovsky iron-

mine, has been opened to a depth of 175 ft. by means of a horse-whim. The ore is a soft rock, almost clay, probably a porphyry. Augite-garnet rock and much quartz are also present. It is worked for a width of 38 ft. in a portion of the mine. The ground is very heavy, and is taken out by running closely-timbered drifts 50 ft. apart. These are allowed to settle after being filled with waste while parallel drifts are being run. Eventually all the ore will be extracted. Mining-costs in 1904 were $1.56 per ton; the ore-reserves were estimated at 68,500 tons, valued at $225,000. The ore is broken by hand with hammers, and worked in a Chilean mill, containing two iron wheels of 55 in. diameter by 11 in. face, and weighing 5,400 lb., which revolve in an iron pan and make 9 rev. per min. They are run by a 12-h.p. engine, and crush 23 tons of quartz or 42 tons of clay in 24 hr. The cost of milling is 48 cents per ton. A 5-stamp mill also ran on ore from this mine. The stamps weighed 800 lb., dropped 8 in. from 80 to 90 times a minute, and crushed 11 tons of quartz or 27 tons of clay in 24 hr. through a screen with 1- by 5-mm. slots, at a cost of 84 cents per ton. For this class of ore and these workmen, the Chilean mill is the best machine for crushing.

By reason of surface-decomposition, outcropping quartz-ledges are not common. Some remarkably rich quartz-float has been found. It occurred in clay, and was used for making brick until it was noticed to be auriferous. From 280 lb. of quartz in one boulder that was yellow with gold, 20 lb. of gold was obtained. Mr. Savitsky, who has charge of the gold-mining of the estate, considered these boulders as lying on the edge of a Tertiary sea. Some barren quartz-veins were found in the vicinity; but nothing of value was developed.

Placer-gold mining has been carried on by dredges as well as hand-labor, and vein-gold mining has been started. An account of the gold-dredging in this section is given in my paper, Gold-Dredging in the Urals.[2] At present (1907) the dredges in this section are doing fairly well. There is room on the estate for a large number of them.

In the gold-placers worked by pick and shovel, the deposits average 14 ft. in depth. The upper layer of "torf" or soft dirt

[2] *Trans.*, xxxvii., 322 to 330 (1907).

contains little gold; the remaining 12 ft. is sand, gravel and clay, the largest stones being 6 in. long. Usually dirt is washed near the placers; but in some cases it is transported several miles by railway. The gravel is dumped on a grating with narrow openings through which it is washed by two 1.5-in. streams supplied by a Worthington pump under from 8 to 9 atmospheres pressure; the theoretical capacity of the pump being 120 cu. ft. per min. under 12 atmospheres. This washing breaks up the clay, and is said to be more effective than the ordinary Siberian machine used for the same purpose. The pebbles which collect on the grating are washed away through a trap-door, opened from time to time. The screenings flow into sluices, where the gold is collected with the aid of mercury. The estimate of costs for 1904 was 36 cents per cu. yd. for all material handled. The washing of the pay-dirt cost 13 cents per cu. yd. These costs were regarded as abnormally high. The actual costs at a placer worked some years ago were 16 cents per cu. yd. for all the material excavated, or 20 cents per cu. yd. for the pay-dirt only.

The estimate of the yield of these placers for 1904, including the dredge, was $205,000 gold and 237 oz. of platinum.

Starateli, or leasers, of whom there are 1,000 at work, are expected to produce $164,500 in gold, which is bought by the estate at $12.25 per ounce, its actual value being $18.60. Considerable expense is incurred in overseeing these leasers, and the profit is not so great as that secured by dredging. Moreover, some of the gold won by the leasers is sold secretly. For these reasons the number of leasers is to be reduced. Fig. 6 is a view of a placer-deposit, showing *starateli* at work.

VIII. Copper.

Deposits of copper-ore have been found in many places on the estate. The average yield of the ores now mined is 5 per cent. It was formerly much higher.

Having devoted most of my time on the estate to the study of the copper-industry, I am able to give on that subject more abundant details, for many of which I am indebted to the kindness of N. L. Gerke, the manager of the smelting-works.

As early as 1750, this region was renowned for its deposits of copper as well as of iron. The copper-mines have been

worked steadily since 1745. The copper blast-furnaces at Bogoslovsk are in a brick building erected more than 100 years ago. Copper-smelting was begun in 1765. From 1896 to 1902 the production was 9,420 tons of copper, at an average cost of 13.6 cents per lb. The works have been enlarged recently; in 1906 the product was 2,300 tons; and it is hoped to make 6,000 tons in 1907. The chief mines are at Turinske Rudnik, 8 miles from Bogoslovsk.

Geology and Mineralogy.—A geological survey of the estate, begun by Professors Feodoroff and Nikitin, has been continued by Professor Stratanovich, a gifted man, who was perhaps fortunate in being exiled to this region while a student in the Royal School of Mines at St. Petersburg. The following brief account of the geology is chiefly due to him.

The oldest rocks seen near the copper-mines are the Lower Devonian limestones. These have been penetrated by many eruptive rocks, the first of which were granites, aplites, and porphyries, mingled with many strata of tuffs. This complex was overlaid by "tentaculite" slate; and, still later, an eruption of augite-garnet rock broke through the whole. This augite-garnet rock was first described by Professor Feodoroff, from its occurrence here. All of the copper-deposits are contact-deposits between augite-garnet and porphyry or limestone.

The chief copper-ore is chalcopyrite. Small quantities of chalcocite, bornite, native copper, cuprite, and a number of other copper-minerals are found. Pyrrhotite and pyrite are abundant; and magnetite is found in large amount in the lower levels of the Bashmakovsky mine, in which the rich copper-minerals of the upper levels have given place to pyrrhotite with 0.5 per cent. of copper, and this, at still greater depth, to magnetite. The chief non-metallic minerals are calcite, garnet, augite, and quartz.

The copper-mines now being worked are Bashmakovsky, Bogoslovsky, Frolovsky, Juravlinsky, Nikitinsky, and Vasilevsky. They are situated in a belt 3 miles in length; and although work has been carried on here for more than 140 years, there still remains a large extent of unexplored ground in the vicinity, and there are a number of promising exposures of copper-ore at other places on the estate.

The Bashmakovsky Mine.—This vein strikes a little west of

north (the general strike of all the deposits), and dips 30° W. The main ore-body, now nearly worked out, was 820 ft. long, with a maximum thickness of 20 ft. and a length, on the dip, of 500 ft. This was a contact-deposit between augite-garnet and hornblende-andesinophyr. The augite-garnet rock, from 130 to 160 ft. thick, formed the foot-wall, underlain by a feldspathic rock, from 13 to 20 ft. thick, and below by another layer of augite-garnet. To the northwest the vein splits up and dies out; and in depth the ore changes to pyrrhotite carrying 0.5 per cent. of copper. The 375-ft. level shows a body of pyrrhotite 420 ft. long and 32 ft. thick. This ore is used for making sulphuric acid in the chemical works, where 14 of its 28 per cent. of sulphur-content is utilized. Below the 375-ft. level, the pyrrhotite contains large amounts of magnetite. On the 448-ft. level, the vein was cut off by a porphyry dike and prospecting with the diamond-drill was unavailing. A narrow vein of gold-quartz was cut on some of the levels, but has not been followed. Manganese-ore is also found here.

The Bogoslovsky Mine.—Nearly half of the copper-ore mined by the company comes from the Bogoslovsky mine. Here there are two veins, having a flat dip to the west. The augite-garnet rock occurs as two flat dikes in porphyry, interstratified with tuffs parallel to the veins. Limestone is found in blocks and also forms the foot-wall of a portion of the lower vein. The augite-garnet dikes are from 13 to 33 ft. thick, and usually 30 ft. apart; but they unite on the 210-ft. level. The "main" vein is from 3 to 20 ft. thick, the "parallel" vein somewhat thinner. The tenor of copper does not decrease with depth, as in the Bashmakovsky mine; but the ore-bodies diminish downward. In 1904, the best ore-body was 70 ft. long by 140 ft. on the pitch, and 14 ft. in thickness, and its average yield was 5 per cent. of copper.

It has been very difficult to follow the ore in this mine because of the faults, both normal and reverse, which have been encountered. On one occasion, the lost vein was sought for with a diamond-drill. The cores were carefully examined by expert microscopists, who reported that the vein had not been found. Cross-cuts, run subsequently by the mine-managers, disclosed the vein, and it was seen that the diamond-drill

had passed through it. When this was reported to the experts they re-studied their rock-sections, and found that the vein-minerals could be recognized microscopically, but that their characters had been masked by epidotization. This interesting experience shows that it is not safe for a mine-manager to rely absolutely upon petrographic conclusions, even when they are supported by high authority. I may add that I have seen elsewhere illustrations of this condition.

The Frolovsky, Juravlinsky, and Nikitinsky Mines.—In these mines the ore occurs as a contact-deposit of steep and irregular dip between augite-garnet rock and limestone. The ore-bodies in the Frolovsky mine vary greatly in size; the largest being 200 ft. long on the pitch, with a maximum cross-section of 1,000 sq. feet.

The Vasilevsky Mine.—In this mine the vein was well defined in the upper levels, but the formation in the lower levels is much broken up, and the ore is in bunches, though richer (8.5 per cent. of copper) than in the other mines. Surface-decomposition, which extends to the 210-ft. level in the other mines, reaches in this mine the 539-ft., or lowest, level.

According to the estimates of the engineers, there were in these mines, at the end of 1904, but 30,000 tons of ore-reserves, or one year's supply for the smelter. But the mines have been running steadily ever since, and are now producing at the rate of 100,000 tons per annum. There is a very large amount of unprospected ground on the estate, and there seems to be no reason why the mines may not be prosperous for a long time to come.

Exploitation.—One of the most remarkable features in the operation of these mines is the very slow progress made in opening new ground. This is partly due to the extreme hardness and toughness of the augite-garnet rock; but in the softer rocks also, progress is slow. In a drift through pure crystalline limestone in the Frolovsky mine, with two machine-drills, and six men in the face for two 8-hr. shifts daily, the advance was but 25 ft. per month; and the average of a number of other drifts in the same mine was 19 ft. per month. The depth of the holes bored was 19 in. By hand-drilling, the advance in limestone drifts with four men, working two 8-hr. shifts daily, was 14 ft. per month. This slow advance is due to poor management, poor tools, inferior labor, and low air-pressure for the drilling-machines. It could easily be bettered. I found in Paris a respon-

sible firm of contractors, who agreed to guarantee an advance of 200 ft. per month in limestone like the sample exhibited to them. German contractors declined to attempt the work, on account of the responsibility incurred by managers under the Russian laws. But they consider the actual rate of progress absurdly slow. R. Meyer, a German contractor of Mülheim, claims a progress for ordinary mine-drifts in trachyte of 465 ft. per month, and in limestone an average of 294 ft. for a number of months.

Mining-Costs and Methods.—In the Bogoslovsk mines, the cost of drifting was $9.40 per linear foot, which compares favorably with that in other countries. That is to say, under the system of payment stated below, the economic loss due to slow progress is chiefly not of direct money-cost, but of time.

Miners are paid by contract, with a guaranteed minimum of $0.33 per diem.

Estimated mining-costs per ton of 2,000 lb. for 1904 were:

Bashmakovsky mine,	$8.93
Bogoslovsky mine,	5.94
Frolovsky mine,	9.01
Vasilevsky mine,	16.50

The classified costs per ton at the Bogoslovsky mine are:

Pumping (average 27 cu. ft. of water per min.),	$0.96
Dead-work,	2.15
Stoping,	1.25
Timbering and filling stopes,	0.15
Tramming,	0.36
Hoisting,	0.24
Repairs,	0.05
Mine-management,	0.34
Administration (general expenses, hospitals, schools, taxes, etc.,)	0.44
	$5.94

The high pumping-costs are largely due to lack of boiler-capacity, and the use of too small and too many pumps; there being, as it were, a museum-collection of pumps at nearly every shaft. This was well known to the local engineers; but they were unable to make needed changes, for lack of funds.

Siemens & Halske electric drills were a failure at the Vasilevsky mine. They "heated" after a few minutes' running.

Very complete plans of the mines are kept up to date by the

engineer corps; in some cases horizontal plans were made for every 7 ft. in depth; but no assay-plans were made of the copper-mines.

The ore is sorted on the surface, by girls and women, in houses built for the purpose, as shown in Fig. 7. The waste rock is seldom assayed; but it is reported to contain an average of 0.5 per cent. of copper.

There are three sorting-houses. The best has a rock-breaker, the ore from which falls over an iron grating with bars 2.5 in. apart. The screened ore falls on a grating with $\frac{5}{8}$-in. openings; the ore remaining on each of these gratings is washed by a stream of water and hand-picked by girls, who are paid by contract and earn an average of 15 cents daily in winter and 30 cents in summer. The material passing the $\frac{5}{8}$-in. screen is saved as ore. In 1903, the cost of sorting varied at the several mines from $0.20 to $0.56 per ton.

Wet-concentration of the ore seems to be impracticable by reason of the large proportion of magnetic pyrites. Experiments in magnetic concentration and by the Elmore process have not yet been successful.

Metallurgical Practice.—Nearly all the ore is roasted in the open air. At the Frolovsky mine this is done in stalls 70 ft. long by 14 ft. wide, with walls 10 ft. high, and provided with flues. No shed or roof covers the ore. Cordwood is laid down first; over this is a layer of coarse ore; then fine ore; then a second layer of wood; then coarser ore; and then fine ore as a cover.

At the Bogoslovsky mine, the ore is roasted in heaps, as shown in Fig. 8, and in stalls. The heaps are truncated pyramids 42 by 50 ft. on the bottom and 7 ft. high, each containing 550 tons of ore. The ground is covered with coarse ore; over this is placed 12.5 cords of wood; then medium-sized ore. The top and sides are covered with fine ore; and cordwood is piled vertically at the four corners for chimneys. The ore is burned for from two to three weeks, then repiled and reburned for an equal time. The consumption of wood equals one-seventh of the volume of the ore.

The Bogoslovsky stalls are 63 ft. long by 28 ft. wide and 7 ft. high. One layer of cordwood is used. The ore is roasted twice, the first roasting taking from four to six weeks, and the

second from three to five weeks. With allowance for cooling, the operation requires from ten to twelve weeks in all. It reduces the percentage of sulphur from 20 to 6.5. The cost of roasting per ton of 2,000 lb. during the year 1903 was:

Bashmakovsky mine,	$0.21
Bogoslovsky mine,	0.10
Frolovsky mine,	0.09

The stall-roasting gives the best results, and would be used exclusively, were stalls enough available.

There has been in the past much friction between the managers of the mines and of the smelting-works, with regard to the copper-content of the ore, because the sample-assays of the mines showed more copper than the smelters were willing to account for. This was partly due to the desire of each department to show as good results as possible. In former years, the difference was sometimes enormous. For example, the mine-assays for a year showed an average of 7 per cent. of copper, while the smelter-assays gave only 4.81 per cent. Such controversies between the different departments of enterprises under one ownership have not been unknown elsewhere.

At Bogoslovsk, the question of sampling was thoroughly investigated, and in 1902 the chief chemist, E. Juon, made a report on the losses in smelting and the differences in assaying, which was in part published in the *Österreichische Zeitschrift für Berg- und Hüttenwesen*. After that report, the difference decreased greatly. In 1903 the mine-assays gave 5.49, and the smelter-assays 4.81 per cent. The introduction of mechanical sampling was thought too expensive by the management, but probably would have been a good investment.

Mr. Juon found that the (roasted) ore was carefully sampled at the mines; that there was some loss in dust; that the copper-content of the ore was determined at the smelter from the assay and weights of the matte and slags, the amount of copper in these two products being taken as the amount received by the smelters. A careful study for 12 days at one furnace showed the amount of copper in the slag, matte, and other smelting-products to be 3.72 per cent. less than in the material fed into the furnace, which loss of 3.72 per cent. must be accounted for by the flue-dust and by volatilization. A similar

study of the Bessemer converter showed a loss of 11.26 per cent. of copper, chiefly in dust, which was produced in excessive quantities by the nearly vertical position of the tuyeres. Samples from the roof of the smelting-house assayed 70 per cent. of copper, and showed globules of copper 0.1 in. in diameter. After this investigation, a flue ending in a brick chamber 36 ft. long by 27 ft. wide and 13 ft. high, and connecting with a stack 125 ft. high, was built, and the loss was greatly reduced. The loss in refining was found by Mr. Juon to be 1.98 per cent.; but this result is doubted by the metallurgical officials.

At present, the ore is carefully sampled by hand-shoveling, both at the mines and at the smelter, and I am convinced that part of the remaining difference of 0.68 (5.49–4.81) per cent., between the mines and the smelter is a real loss, due in part to the leaching of the roasted ore by rains (the rainfall is 15 in.) and partly to loss in dust. No doubt there is a remainder, the cause of which may be called "psychological." The question of losses due to leaching by the rain caused trouble at Ducktown, Tenn., as Dr. Peters relates in his valuable work on the metallurgy of copper.

Copper-smelting was begun in Bogoslovsk in 1765, in which year 722 tons were worked. Water-power supplemented by steam is used, the furnaces taking 40 h.p., the converters 130 horse-power.

In August, 1904, there were: four Pilz furnaces 48 in. in diameter, each having a daily capacity of 18 tons; three Raschette furnaces, one, 144 by 42 in., daily capacity 54 tons, and two, 65 by 42 in., daily capacity 14 tons each. The annual capacity was 36,000 tons of ore. Since then the smelter has been enlarged. A furnace 375 by 42 in. in size has been built; and it is hoped to smelt more than 100,000 tons of ore to yield 6,000 tons of copper during 1907.

The furnaces are of local brick. Charcoal is used for fuel, and up to 1906 it had been found too soft to use in a water-jacket furnace with high blast-pressure. In 1905, the pressure was 160 mm. of water (3.75 oz. per sq. in.).

The ordinary furnace-charge is:

Bogoslovsk mine ores,	36,000 lb.
Custom ores,	1,800 lb.
Slags,	3,600 lb.
Burnt lime,	1,800 lb.

In 1903, the smelter treated 28,476 tons of Bogoslovsk ores, 593 of custom ores, 1,133 of soil from the smelter-yard, and 265 of converter-slag. The product was 5,071 tons of matte, making 1,445 tons of blister copper, or 1,374 tons of refined copper.

The matte and slag run continuously into a fore-hearth; the slag overflowing into running water, whence it is shoveled into railway-cars and taken away to be used as railway-ballast. The matte is intermittently tapped, in 2-ton charges, into a clay-lined pot mounted on wheels and hauled by a horse to a horse-capstan, which hoists the pot to the converter-floor. The smelting-costs per ton (2,000 lb.) of ore were, in 1903: labor, $0.44; charcoal, $1.16; repairs and supplies, $0.52; total, $2.12.

The laborers are paid on a complicated system, the growth of more than a century. The furnace-men get 14 cents per ton or, for an average day, $3.25, divided among two furnace-men and four helpers. Charcoal to the amount of 24.33 per cent. of the weight of the charge is allowed, and 18 per cent. of the value of the charcoal saved on this basis is paid as a premium. The actual proportion of charcoal used is about 23 per cent. The other employees are paid on similar systems. Day-laborers receive from 25 to 30 cents daily.

The matte varies in copper from 18 to 50 per cent. In 1903, it averaged 27.11 per cent. There is no trouble in keeping the low-grade matte liquid during its ten minutes' passage to the converters; but the 40 to 50 per cent. matte gives trouble by chilling.

There are eight converters, two of which are in blast at a time. These were introduced in 1885, and represent the Manhès converter, considerably modified to avoid paying a royalty. This was a mistake; for the upward-blowing tuyeres occasion a very large loss in copper. The blast-pressure is 280 mm. of mercury (87 oz. per sq. in.).

The converter-shells, 6 ft. in diameter, are lined with a mixture of 80 per cent. of quartz and 20 per cent. of clay—the latter carrying some coarse quartz. For each pound of copper made, 1.2 lb. of lining is consumed. The lining is put in as soon as the shell cools down to 170° F., this being the maximum temperature the workmen will endure. The lining is 1 ft. thick, so that the newly-lined converter has an inside diameter of 4 ft.

An ideal heat would be: 3,600 lb. matte charged in converter; blast for 40 min.; 3,600 lb. of slag poured off, leaving in the converter 1,440 lb. of white metal containing 70 per cent. of copper, to which a new charge of 3,600 lb. of matte is added; blast for 40 min.; 3,600 lb. of slag poured off, leaving in the converter 2,880 lb. of white metal; blast for one hour; 1,800 lb. of blister copper poured.

The converter has thus received 7,200 lb. of 30-per cent. matte, containing, roughly, 2,160 lb. of copper, and has given: 7,200 lb. of slag with 2.4 per cent. of copper (173 lb. of copper); 360 lb. of flue-dust, 70 per cent. of copper (252 lb. of copper); 1,800 lb. of blister copper, 96.4 per cent. of copper (1,735 lb. of copper), giving a total of 9,360 lb. of material treated containing 2,160 lb. of copper.

There is a gain in weight of 9,360 — 7,200 = 2,160 lb. This comes from the lining, of which 1,624 tons were consumed in 1903 in making 1,374 tons of copper. The notable features in this converting are the low grade of the matte (at times containing but 18 per cent. of copper) and the great consumption of lining.

Two heats are made to each relining of the converter; one lining suffices for the conversion of 14,000 lb. of matte. The above ideal heat is rarely attained in practice. Much white metal goes into the slag, which is poured into large iron pots, and, after breaking up, is sorted into white metal (which is melted in small reverberatory furnaces), clean slag containing 0.3 per cent. of copper (which goes on the dump), and foul slag (which goes to the blast-furnace). In 1903, the product of blister copper was 1,445 tons, which averaged 98 per cent. of copper, and was refined in reverberatory furnaces. Electric refining has been tried and abandoned, but is to be tried again.

In 1903, the converting-cost per 2,000 lb. of matte was:

Repairs,	$0.90
Wood,	0.82
Labor,	1.35
Supplies,	0.70
Power,	0.38
Total,	$4.15

The refined copper is cast in 28-lb. ingots. The cost of refining the blister copper is $4 per ton.

The total costs for 1903 were:

	Per 2,000 Lb. of Ore.	Per Pound of Copper. Cents.
Mining, sorting, and roasting,	$8.59	9.06
Matte-smelting,	2.12	2.20
Refining,	0.20	0.21
Converting,	0.89	0.92
Management,	0.33	0.34
Administration,	0.95	1.05
	$13.08	13.78

Comparison with costs in the United States shows that the high cost of the Bogoslovsk copper is due mainly to mining. The management- and administration-costs are also high,—the latter necessarily so, by reason of legal requirements, which oblige the estate to do work elsewhere done by county and State governments.

Stevens[3] gives the cost per ton of ore treated during 1905, by the Tennessee Copper Co. at Ducktown, Tenn., as follows:

Mining,	$0.686
Development,	0.171
Crushing and sorting,	0.106
Transportation,	0.128
Matte-smelting,	1.24
Converting,	0.2313
Sundry,	0.208
	$2.7703

This copper was not refined. Considering the small quantity of ore handled by the Bogoslovsk estate, the smelting-costs do not seem high, and they will be considerably reduced in the enlarged plant, now in operation.

The following analyses of the copper-ores and products will be found interesting:

Analyses of Copper-Ores.

	SiO_2.	Al_2O_3.	Fe.	Cu.	S.	CaO.	MgO.	Mn.
Bashmakovsky, average for 1901, raw ore,	31.52	1.93	24.96	4.87	18.64	14.18	2.60	0.66
Nikitinsky raw ore,	37.68	2.14	20.96	3.64	14.43	20.22	.....	0.49
Frolovsky raw ore,	33.22	1.11	16.03	7.04	15.82	22.23	4.10	0.46
Bogoslovsky raw ore,	29.08	3.13	23.10	5.87	25.30	9.76	3.21	0.32
Vasilevsky raw ore,	38.15	7.53	18.52	10.19	14.86	8.09	2.17	.....
Roasted ores, Bogoslovsky, 1st class,	27.94	7.52	23.54	4.02	5.4	10.35	4.78	.....
Bogoslovsky, 2d class,	31.96	8.16	22.54	2.39	5.8	11.58	2.16	.....
Frolovsky,	26.59	0.95	16.54	5.45	5.3	24.24	7.81	.....
Bashmakovsky,	25.14	7.95	20.90	5.04	6.0	16.78	2.90	.....
General average, June–Dec., 1903,	27.07	6.82	21.82	5.53	6.57	14.39	1.59	.....
Slag from ore-smelting,	41.10	5.11	18.22	0.12	0.10	15.65	.....	.....

[3] *Copper Hand-Book* for 1907, p. 1075.

Analyses of Matte and Refined Copper.

	Matte. Per Cent.	Refined Copper. Per Cent.
Copper,	31.26	99.66
Iron,	38.91	0.021
Sulphur,	26.69	0.014
Cobalt,	0.51	.....
Nickel,	0.028	0.052
Arsenic and antimony,	trace	0.049
Bismuth,	0.00	0.000
Silver,	0.006	0.040
Lead,	0.008	0.005
Silica,	0.016	0.030 (Tin oxide and silica).
CaO,	0.68	trace of gold.
MgO,	0.12	

IX. Other Mineral Resources.

Since 1805, about 10,000 tons of chrome iron-ore have been mined on the estate, at a cost of $11 per ton. This ore is used in the chemical works for making sodium and potassium bichromate.

Lignite-beds cover many square miles of the estate. A little work has developed 3,600,000 tons. In 1903, there were mined 32,000 tons, at a cost of $0.93 per ton. As a fuel, one ton of this lignite is said to be equivalent to 0.85 cord of wood.

Manganese-ores are produced in small quantity.

X. Russian Weights, Measures, and Money.

I have encountered much difficulty in getting correct equivalents for Russian weights and measures. Their system is as absurd as our own, and makes one long for the day when the metric system shall replace all others.

Some of the values given below have been taken from a paper by A. L. Simon on Siberian Mines and Mining Conditions,[4] and the discussion thereof by Dr. F. H. Hatch, who furnished more accurate determinations of several values.

Weights.

One pood = 40 funt (Pfund) = 3,840 zolotniks = doli 368,640.
= 36.112808327 avoirdupois pounds.
= 526.6451214 Troy ounces.
= 16.3804964 kilograms.

[4] *Transactions of the Institution of Mining and Metallurgy*, vol. xvi., pp. 358, 382 (1906–07).

One pood of pure gold = 21,157.025 rubles.
= $10,886.718.
One funt = 96 zolotniks = 9,216 doli.
= 409.51241 grams.
= 13.166128 Troy ounces.
= 0.902820208 avoirdupois pound.
One funt of pure gold = $272.17.
One zolotnik = 96 doli.
= 0.1371471 Troy ounce.
= 4.26575 grams.
One zolotnik of pure gold = $2.83508.
One dolya = 0.0014286 Troy ounce.
= 0.0444349 gram.
One dolya of pure gold = $0.029532.
One 2,240-lb. ton = 62.02783 poods.
One 2,000-lb. ton = 55.38201 poods.
One pood = 0.0180564 ton of 2,000 lb.
One ton (2,000 lb.) of pure gold = $602,928.375.
One Troy ounce of pure gold = $20.67183.
One ruble = $0.514567.

According to the British "mint parity value" of 25.375 pence, given by Dr. Hatch, and taking the value of the Troy ounce of gold as £4 4s. 11.454d., and the value of the pound sterling as $4.866565, the gold ruble would be worth $0.514538.

The value I have given above was calculated from the statement on the 10-ruble gold coin, that it contains 1 zolotnik and 78.24 doli of gold. The difference of $0.000029 between the two results is of no practical importance.

I would call attention here to several values in the above statement, which are carried to many places of decimals. These have been quoted from Dr. Hatch; but I cannot forbear to point out that they present, in my judgment, a misleading appearance of minute accuracy. For example, the pood is said to be equal to 36.112808327 lb. avoirdupois. Now, the most accurate weighing of our chemical balances will probably determine 1 gram to within one five-hundredth part of a milligram, or 0.000002 g.; that is to say, it would leave the sixth decimal in doubt. It is very easy, of course, to carry out a quotient in division to any number of decimals; but when neither divisor nor dividend can possibly be accurate beyond five places,

spinning out the quotient to eleven places is neither scientific nor significant. All that can be truly said concerning Dr. Hatch's pood is that it is equal to 36.11281 lb., with some doubt as to the last decimal figure, and that the addition of six, or sixty, further figures would not in any degree affect that doubt. The same criticism applies to the data upon which his calculation was founded, such as the solemn official declaration by the International Bureau of Weights and Measures that 1 lb. avoirdupois = 0.4535924277 k., in which result the last digit represents so many ten-thousandths of a milligram! Since this result is the quotient from two magnitudes, neither of which could be accurately determined within two-thousandths of a milligram, it must be regarded as a case of very "long division," and nothing more!

Values Per Cubic Unit.

One ruble per cubic sagene = $0.04053 per cu. yd.
One ruble per pood = $28.50 per 2,000-lb. ton.
= $0.01425 per lb.

Values Per Unit of Weight.

Zolotniks Per 100 Poods.	Equivalent in U. S. Money Per Ton of 2,000 Lb.	Doli Per 100 Poods.	Equivalent in Cents Per Ton of 2,000 Lb.
1,	$1.57	1,	1.64
2,	3.14	2,	3.27
3,	4.71	3,	4.91
4,	6.28	4,	6.54
5,	7.85	5,	8.18
6,	9.42	6.	9.81
7,	10.99	7,	11.45
8,	12.56	8,	13.08
9,	14.13	9,	14.72
10,	15.701	10,	16.355

Lengths and Distances.

One verst = 500 sagenes = 3,500 ft. (The Russian and the English foot are equal.)
= 0.6628786 mile.
= 1.0668 kilometers.
One sagene = 7 ft. = 3 archines.
= 2.1336 meters.
One archine = 16 vershoks.
= 2 ft. 4 in.
= 0.7112 meter.

One vershok = 1.75 in. (The Russian and the English inch are equal.)
= 0.04445 meter.

Areas.

One square verst = 281.2213 acres.
= 0.4394 sq. mile.
= 113.80 hectares.
One dessiatine = 2,400 sq. sagenes.
= 2.6997 acres.
= 1.09252 hectares.
One square sagene = 5.4444 sq. yards.
= 4.55217 sq. meters.

Volumes.

One cubic sagene = 343 cu. ft.
= 12.704 cu. yards.
= 2.68 cords.
= 9.71242 cu. meters.
One chetverik = 26.24 liters.
One vedro = 12.30 liters.

The following illustrations, Figs. 1 to 8, engraved from photographs taken by me, although they may have no special professional value, will serve to give to those who have never visited this part of Russia a notion of the scenery, and of the people who have been for more than a century engaged in developing and utilizing the mineral wealth of an almost unknown corner of the world.

FIG. 1.—HOUSE OF GENERAL MANAGER, BOGOSLOVSK ESTATE.

FIG. 2.—BOGOSLOVSK, SEEN FROM SMELTER.

FIG. 3.—MINING STUDENTS.

FIG. 4.—IRON BLAST-FURNACE AT NADESDA.

FIG. 5.—HAULING IRON-ORE FROM AUERBACH MINE.

FIG. 6.—STARATELI, WORKING A PLACER.

FIG. 7.—ORE-SORTER, BASHMAKOVSKI MINE.

FIG. 8.—HEAP-ROASTING, BOGOSLOVSK MINE.

The Wilfley Table, II.

BY ROBERT H. RICHARDS, BOSTON, MASS.

(New York Meeting, February, 1908.)

My first paper, read at the Cobalt Meeting of the Institute,[1] July, 1907, dealt with the behavior of a small Wilfley table when concentrating galena from quartz, the table being fed with natural products, with sized products, and with classified products. Since galena, having a specific gravity of 7.5, is among the heaviest minerals ordinarily concentrated by Wilfley table, it was desirable that a similar set of tests upon a mineral of lighter weight should be made. Chalcopyrite, of a specific gravity of 4, would have been the best mineral for this purpose, on account of its great importance as an economic mineral, but in carrying out the test I was obliged to content myself with a cupriferous pyrite, the specific gravity of which was 4.68 and the content in copper 8.80 per cent. Through the kindness of H. O. Cummins, this material was obtained from Shasta, Cal., in very pure, clean, homogeneous condition. It was broken by rolls to 2 mm. in size, taking care to make a minimum of slimes. The quartz used to mix with it was pure, white, New England quartz, broken in the same way to the same size.

In studying the accompanying tables, it should be borne in mind that the material concentrated was of low copper-content, and in many cases the quantity fed to the machine was 1 kg. or less. As a consequence of the smallness of the quantity treated it was very difficult to adjust the table and get it in condition to do its best work before the feed became exhausted, and for this reason many results were obtained which appear irregular.

[1] *Trans.*, xxxviii., 556 to 580 (1908).

The material concentrated on the Wilfley table was a mixture of quartz and cupriferous pyrite; 16 test-runs were made. Runs Nos. 1 to 4 were on natural products of the following sizes: 2 mm. to 0, 1 mm. to 0, 0.5 mm. to 0, and 0.25 mm. to 0, respectively. Runs Nos. 5 to 10 were on sized products.

Table I. gives the sizes and actual weights of material fed to the Wilfley table.

TABLE I.—*Sizes and Weights of Material Fed to the Wilfley Table, Tests Nos. 5 to 10.*

Run Number.	Sieve-Diameter.		Actual Weight Fed.
	Through.	On.	
	mm.	mm.	kg.
5	2	1.4	18.000
6	1.4	1.0	11.100
7	1	0.75	8.600
8	0.75	0.50	7.800
9	0.50	0.36	3.475
10	0.36	0.28	1.060
Not run.	0.28	0.00	4.965

The total weight of 55 kg. given in Table I. was made up of 50 kg. of quartz and 5 kg. of pyrite.

Tests 11 to 16 were on classified products. For rising-currents, see my first paper[2] on the Wilfley Table. The classifier there described was used.

TABLE II.—*Results of Tests Nos. 1 to 4 on Natural Products, 100-Ton Basis.*

	Run No. 1.	Run No. 2.	Run No. 3.	Run No. 4.
Size of feed	mm. 2 to 0	mm. 1 to 0	mm. 0.5 to 0	mm. 0.25 to 0
Rate of feed per 24 hr.	Tons. 22	Tons. 22	Tons. 11	Tons. 11
Concentrates	2.877	6.006	6.231	6.519
Middlings	47.957	29.024	23.179	9.366
Tailings	48.050	62.849	64.360	75.070
Slimes	1.116	2.121	6.230	9.045
Total	100.000	100.000	100.000	100.000

[2] *Trans.*, xxxviii., Table III., p. 562 (1908).

TABLE III.—*Results of Tests Nos.* 5 *to* 10 *on Sized Products,* 100-*Ton Basis.*

	Run No. 5.	Run No. 6.	Run No. 7.	Run No. 8.	Run No. 9.	Run No. 10.
Size of feed	mm. 2 to 1.4	mm. 1.4 to 1	mm. 1 to 0.75	mm. 0.75 to 0.50	mm. 0.50 to 0.36	mm. 0.36 to 0.28
Rate of feed, 24 hr.	Tons. 22	Tons. 22	Tons. 22	Tons. 22	Tons. 11	Tons. 11
Concentrates	10.704	5.824	5 245	5.097	7.524	9.519
Middlings	5.046	3.160	3.454	3.923	8.449	3.461
Tailings	84.250	91.016	91.301	90.980	84.027	87.020
Slimes	0.000	0.000	0.000	0.000	0.000	0.000
Total	100.000	100.000	100.000	100.000	100.000	100.000

TABLE IV.—*Results of Tests Nos.* 11 *to* 16 *Computed to Full-Sized Table and* 100-*Ton Lots.*

	Run No. 11.	Run No. 12.	Run No. 13.	Run No. 14.	Run No. 15.	Run No. 16.
Classifier spigot numbers fed	1 + 2	3 + 4	5 + 6	7 + 8	9 + 10	11 + 12
Classifier currents, mm. per sec. through which grains settled	85	55.4	36.3	23.6	15.4	10
Rate of feed to Wilfley per 24 hr.	Tons. 22	Tons. 22	Tons. 22	Tons. 11	Tons. 11	Tons. 11
Concentrates, tons	41.756	5.516	4.696	5.810	8.469	10.606
Middlings, tons	5.178	5.336	3.790	5.213	8.650	3.413
Tailings, tons	53.066	89.148	91.514	88.977	82.881	85.981
Slimes, tons	0.000	0.000	0.000	0.000	0.000	0.000
Total	100.000	100.000	100.000	100.000	100.000	100.000

A comparison of the results of a sized and classified feed with those of an unsized product (natural feed) is of interest. Table V. shows that both sized feed and classified feed give better results than natural feed in the percentage of copper which is easily and quickly saved in the concentrates. Furthermore, the tailings from the sized product are decidedly better than those from the classified product, although the latter do not carry a large percentage of the copper, and the tailings from both of these classes are better than those from the natural feed.

TABLE V.—*Summary of Results of Tests Nos.* 1 *to* 16.

No. of Run.	Size.	Rate of Feeding. Kg. Per Min.	Rate of Feeding. Tons per 24 hrs. Large Table.	Quantity Fed.	Composition of Feed. Copper.	Percentage of the Products. Concentrates.	Middlings.	Tailings.	Slimes.	Composition of the Products. Percentage of Copper. Concentrates.	Middlings.	Tailings.	Slimes.	Percentage of Total Copper. Concentrates.	Middlings.	Tailings.	Slimes.
	mm.			Kg.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.
1	2 to 0.	1	22	11	0.88	2.877	47.957	48.050	1.116	5.01	1.10	0.089	2.97	19.27	70.57	5.73	4.43
2	1 to 0.	1	22	11	0.88	6.006	29.024	62.849	2.121	6.09	1.02	0.10	2.82	46.62	37.74	8.01	7.63
3	0.5 to 0.	0.5	11	11	0.88	6.231	23.179	64.360	6.230	5.60	0.76	0.16	1.56	48.11	24.29	14.20	13.40
4	0.25 to 0.	0.5	11	11	0.88	6.519	9.366	75.070	9.045	5.12	0.96	0.18	1.48	48.12	12.96	19.62	19.30
5	2 to 1.4.	1	22	18	0.98	10.704	5.046	84.250	.....	7.66	3.25	0.00	.....	78.88	21.12	0.00	.....
6	1.4 to 1.	1	22	11.1	0.52	5.824	3.160	91 016	.....	7.66	2.26	0.00	.....	86.21	13.79	0.00	.....
7	1 to 0.75.	1	22	8.6	0.45	5.245	3.454	91.301	.....	7.77	1.34	0.00	.....	89.75	10.25	0.00	.....
8	0.75 to 0.5.	1	22	7.8	0.41	5.097	3.923	90 980	.....	6.36	2.24	0.00	.....	78.68	21.32	0.00	.....
9	0.5 to 0.36.	0.5	11	3.475	0.67	7.524	8.449	84.027	.....	6.35	2.33	0.00	.....	70.78	29.22	0.00	.....
10	0.36 to 0.28.	0.5	11	1.060	0.67	9.519	3.461	87.020	.....	6.82	0.27	0.00	.....	98.68	1.32	0.00	.....
	Classifier, Spigots.																
11	1 and 2.	1	22	16.4	3.51	41 756	5.178	53.066	.....	7.83	4.56	0.017	.....	93.02	6.72	0.26	.....
12	3 and 4.	1	22	16.6	0.41	5.516	5 336	89.148	.....	5.03	2.18	0.013	.....	67.73	28.39	3.88	.....
13	5 and 6.	1	22	10.5	0.37	4 696	3 790	91.514	.....	6.30	1.62	0.016	.....	79.56	16.51	3.93	.....
14	7 and 8.	0.5	11	3.075	0.45	5.810	5 213	88.977	.....	6.35	1.39	0.008	.....	82.25	16.17	1.58	.....
15	9 and 10.	0.5	11	2.275	0.68	8.469	8.650	82.881	.....	6.51	1.25	0.021	.....	81.46	15.97	2.57	.....
16	11 and 12.	0.5	11	2.625	0.72	10.606	3.413	85.981	.....	5.53	2.99	0.036	.....	81.53	14.19	4.28	.....

TABLE VI.—*Results of Tests Nos.* 1 *to* 4 (*Concentrates*).

Concentrates sized and weighed. The weights are computed in tons and fractions of a ton on the 100-ton basis.

Sizes.		Run No. 1.	Run No. 2.	Run No. 3.	Run No. 4.
Through.	On.				
mm.	mm.	Tons.	Tons.	Tons.	Tons.
.....	2.06	.....	.....	.....	.....
2.06	1.63	0.002	.....	.....	.....
1.63	1.44	0.008	.....	.....	.....
1.44	1.27	0.012	.....	.....	.....
1.27	1.10	0.019	.....	.....	.....
1.10	0.97	0.026	0.102	.....	.....
0.97	0.84	0.088	0.299	.....	.....
0.84	0.68	0.099	0.422	.....	.....
0.68	0.57	0.157	0.499	.....	.....
0.57	0.45	0.293	0.839	0.581	.....
0.45	0.36	0.280	0.442	0.805	.....
0.36	0.28	0.536	0.801	0.957	.....
0.28	0.24	0.302	0.389	0.628	0.428
0.24	0.20	0.411	0.680	0.513	1.356
0.20	0.15	0.204	0.402	0.728	1.058
0.15	0.12	0.185	0.384	0.632	1.038
0.12	0.10	0.059	0.159	0.305	0.418
0.10	0.08	0.136	0.329	0.608	0.996
0.08	0.00	0.060	0.259	0.474	1.225
Total ...	.....	2.877	6.006	6.231	6.519

Analyses of the above products giving percentages of copper.

Sizes.		Run No. 1.	Run No. 2.	Run No. 3.	Run No. 4.
Through.	On.				
mm.	mm.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
.....	2.06	.....	.....	.....	.....
2.06	1.63	4.40	.....	.....	.....
1.63	1.44	7.92	.....	.....	.....
1.44	1.27	8.54	.....	.....	.....
1.27	1.10	8.60	.....	.....	.....
1.10	0.97	8.68	8.80	.....	.....
0.97	0.84	8.76	8.80	.....	.....
0.84	0.68	8.33	8.80	.....	.....
0.68	0.57	8.35	8.80	.....	.....
0.57	0.45	6.80	7.81	7.03	.....
0.45	0.36	6.60	6.52	6.97	.....
0.36	0.28	3.76	5.72	6.26	.....
0.28	0.24	3.31	3.81	6.43	6.71
0.24	0.20	3.12	3.68	5.39	6.01
0.20	0.15	3.94	4.15	4.99	6.53
0.15	0.12	3.17	5.08	4.57	6.11
0.12	0.10	4.37	5.51	5.38	6.19
0.10	0.08	4.85	5.11	4.10	5.53
0.08	0.00	3.54	3.03	2.84	0.84

TABLE VII.—*Results of Tests Nos.* 1 *to* 4 (*Middlings*).

Middlings sized and weighed. The weights computed in tons and fractions of a ton on the 100-ton basis.

On.	Run No. 1.	Run No. 2.	Run No. 3.	Run No. 4.
mm.	Tons.	Tons.	Tons.	Tons
2.06	0.194	.....	.....	.....
1.63	5.361	.....	.....	.....
1.44	3.891	.....	.....	.....
1.27	4.010	.....	.....	.....
1.10	3.723	.....	.....	.....
0.97	2.450	0.960	.....	.....
0.84	5.112	1.638	.....	.....
0.68	4.178	2.152	.....	.....
0.57	3.801	2.085	.....	.....
0.45	5.474	5.136	1.577	.....
0.36	2.708	3.355	1.970	.....
0.28	3.267	5.175	4.405	.....
0.24	1.081	2.417	3.230	0.639
0.20	1.234	2.726	3.155	1.840
0.15	0.717	1.260	3.566	1.371
0.12	0.248	0.789	2.300	2.378
0.10	0.134	0.447	0.749	0.777
0.08	0.184	0.564	1.306	1.107
0.00	0.190	0.320	0.921	1.254
Total.....	47.957	29.024	23.179	9.366

Analyses of the above products giving percentages of copper.

On.	Run No. 1.	Run No. 2.	Run No. 3.	Run No. 4.
mm.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
2.06	2.84	.....	.....	.....
1.63	2.95	.....	.....	.....
1.44	2.66	.....	.....	.....
1.27	1.86	.....	.....	.....
1.10	1.24	.....	.....	.....
0.97	0.93	6.38	.....	.....
0.84	0.80	5.14	.....	.....
0.68	0.50	2.99	.....	.....
0.57	0.50	0.76	.....	.....
0.45	0.25	0.67	3.56	.....
0.36	0.13	0.07	1.77	.....
0.28	0.09	0.02	0.42	.....
0.24	0.09	0.02	0.35	0.62
0.20	0.03	0.04	0.19	0.43
0.15	0.06	0.10	0.12	0.48
0.12	0.12	0.81	0.10	0.38
0.10	1.76	0.61	0.24	0.63
0.08	2.69	1.10	0.69	0.75
0.00	4.03	4.24	3.55	3.96

TABLE VIII.—*Results of Tests Nos. 1 to 4 (Tailings).*

Tailings sized and weighed. The weights computed in tons and fractions of a ton on the 100-ton basis.

On.	Run No. 1.	Run No. 2.	Run No. 3.	Run No. 4.
mm.	Tons.	Tons.	Tons.	Tons.
2.06	2.992	.....	.....	.....
1.63	15.848	.....	.....	.....
1.44	8.620	.....	.....	.....
1.27	7.029	.....	.....	.....
1.10	2.859	.....	.....	.....
0.97	2.558	8.248	.....	.....
0.84	1.257	14.752	.....	.....
0.68	0.754	9.868	.....	.....
0.57	0.336	5.594	.....	.....
0.45	0.363	8.984	18.650	.....
0.36	0.224	3.223	8.839	.....
0.28	0.351	2.790	13.483	.....
0.24	0.237	0.813	3.506	17.344
0.20	0.853	1.045	5.052	16.603
0.15	0.558	1.728	3.373	6.278
0.12	0.763	1.932	2.372	24.311
0.10	0.367	1.066	2.533	4.174
0.08	1.125	1.554	3.775	4.727
0.00	0.956	1.252	2.777	1.633
Total.....	48.050	62.849	64.360	75.070

Analyses of the above products giving percentages of copper.

On.	Run No. 1.	Run No. 2.	Run No. 3.	Run No. 4.
mm.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
2.06	0.00	.....	.....	.....
1.63	0.00	.....	.....	.....
1.44	0.00	.....	.....	.....
1.27	0.00	.....	.....	.....
1.10	0.00	.....	.....	.....
0.97	0.00	0.00	.....	.....
0.84	0.001	0.00	.....	.....
0.68	0.00	0.00	.....	.....
0.57	0.00	0.00	.....	.....
0.45	0.01	0.01	0.00	.....
0.36	0.02	0.10	0.00	.....
0.28	0.03	0.10	0.32	.....
0.24	0.05	0.05	0.06	0.00
0.20	0.10	0.21	0.05	0.00
0.15	0.13	0.13	0.15	0.11
0.12	0.26	0.31	0.15	0.35
0.10	0.07	0.43	0.15	0.34
0.08	0.73	0.54	0.34	0.36
0.00	2.97	2.59	1.29	0.92

TABLE IX.—*Results of Tests Nos.* 1 *to* 4 (*Slimes*).

Slimes sized and weighed. The weights are computed in tons and parts of a ton on the 100-ton basis.

Sizes.		Run No. 1.	Run No. 2.	Run No. 3.	Run No. 4.
Through.	On.				
mm.	mm.	Tons.	Tons.	Tons.	Tons.
0.57	0.45	.....	.....	0.050	....
0.45	0.36	0.007	.....	0.060	.....
0.36	0.28	0.016	.....	0.116	.....
0.28	0.24	0.009	0.010	0.050	0.023
0.24	0.20	0.019	0.053	0.082	0.076
0.20	0.15	0.019	0.041	0.066	0.193
0.15	0.12	0.070	0.140	0.164	0.335
0.12	0.10	0.134	0.098	0.167	0.306
0.10	0.08	0.290	0.402	1.003	2.001
0.08	0.00	0.552	1.377	4.472	6.111
Total	.....	1.116	2.122	6.230	9.045

Analyses of the above products giving percentages of copper.

Through.	On.				
mm.	mm.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
0.57	0.45	.....	.....	0.45	.....
0.45	0.36	2.16	.....	1.03	.....
0.36	0.28	1.29	.....	1.27	.....
0.28	0.24	1.84	1.73	1.17	1.06
0.24	0.20	1.18	1.59	1.21	0.93
0.20	0.15	1.25	1.19	1.54	1.17
0.15	0.12	1.33	1.08	1.47	0.95
0.12	0.10	2.10	0.69	1.21	0.86
0.10	0.08	2.34	1.65	0.87	0.69
0.08	0.00	3.92	3.60	1.78	1.83

TABLE X.—*Results of Tests Nos.* 5 *to* 10 (*Products*).

Weights computed in tons and parts of a ton on the 100-ton basis.

	Run No. 5.	Run No. 6.	Run No. 7.	Run No. 8.	Run No. 9.	Run No. 10.
	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.
Concentrates.....	10.704	5.824	5.245	5.097	7.524	9.519
Middlings.....	5.046	3.160	3.454	3.923	8.449	3.461
Tailings	84.250	91.016	91.301	90.980	84.027	87.020
Slimes.....	0.000	0.000	0.000	0.000	0.000	0.000
Total	100.000	100.000	100.000	100.000	100.000	100.000

Analyses of the above products giving percentages of copper.

	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Concentrates.....	7.66	7.66	7.77	6.36	6.35	6.82
Middlings.....	3.25	2.26	1.34	2.24	2.33	0.27
Tailings.....	0.00	0.00	0.00	0.00	0.00	0.00
Slimes.....	.....	.....	.....	.....	.....	.....

TABLE XI.—*Results of Tests Nos.* 11 *to* 16 (*Concentrates*).

Concentrates sized and weighed. The weights computed in tons and fractions of a ton on the 100-ton basis.

Sizes.		Run No. 11.	Run No. 12.	Run No. 13.	Run No. 14.	Run No. 15.	Run No. 16.
Through.	On.						
mm.	mm.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.
.....	2.06	0.542	.....	.....	.....	.....	.....
2.06	1.63	9.674	0.0005	.....	.....	.....	.....
1.63	1.44	8.003	0.0009	.....	.....	.....	.....
1.44	1.27	5.232	0.002	0.0002	.....	.....	.....
1.27	1.10	3.018	0.004	0.0003	.....	.....	.....
1.10	0.96	4.173	0.013	0.0007	.....	.....	.....
0.96	0.84	3.244	0.108	0.004	.....	.....	.....
0.84	0.68	1.940	0.379	0.008	.....	.....	.....
0.68	0.57	2.453	0.540	0.057	.....	.....	.....
0.57	0.45	2.133	1.550	0.250	0.003	.....	.....
0.45	0.36	0.550	0.809	0.464	0.010	0.001	.....
0.36	0.28	0.399	1.129	1.157	0.247	0.015	0.011
0.28	0.24	0.130	0.354	0.686	0.198	0.048	0.031
0.24	0.20	0.081	0.399	1.003	1.186	0.344	0.293
0.20	0.15	0.061	0.112	0.593	1.527	1.087	0.720
0.15	0.12	0.030	0.078	0.269	1.097	1.769	1.613
0.12	0.10	0.022	0.008	0.113	0.769	1.447	1.296
0.10	0.08	0.032	0.019	0.068	0.623	2.248	4.018
0.08	0.00	0.039	0.011	0.023	0.150	1.510	2.624
Total.....	.....	41.756	5.516	4.696	5.810	8.469	10.606

Analyses of the above products giving percentages of copper.

On.	Run No. 11.	Run No. 12.	Run No. 13.	Run No. 14.	Run No. 15.	Run No. 16.
mm.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
2.06	8.80	.....	.....	.....	.....	.....
1.63	8.76	6.00	.....	.....	.....	.....
1.44	8.71	6.10	.....	.....	.....	.....
1.27	8.55	7.61	8.80	.....	.....	.....
1.10	8.32	7.62	8.80	.....	.....	.....
0.96	7.78	8.42	8.80	.....	.....	.....
0.84	6.41	8.72	8.44	.....	.....	.....
0.68	6.17	8.70	8.68	.....	.....	.....
0.57	6.22	8.73	8.74	.....	.....	.....
0.45	5.11	7.32	7.94	4.86	.....	.....
0.36	4.54	5.49	7.21	lost.	0.97	.....
0.28	5.37	4.72	8.02	7.61	8.34	2.75
0.24	4.90	4.30	5.72	6.90	8.86	4.90
0.20	4.31	4.98	4.47	6.15	8.38	6.26
0.15	5.96	5.48	5.48	6.94	7.47	6.86
0.12	6.00	6.09	5.51	6.27	7.10	7.00
0.10	6.29	6.34	5.62	5.92	6.85	7.27
0.08	6.40	5.62	6.70	5.53	6 17	6.63
0.00	4.97	7.82	9.71	5.37	4.78	5.48

TABLE XII.—*Results of Tests Nos.* 11 *to* 16 (*Middlings*).

Middlings sized and weighed. The weights computed in tons and fractions of a ton on the 100-ton basis.

Sizes.		Run No. 11.	Run No. 12.	Run No. 13.	Run No. 14.	Run No. 15.	Run No. 16.
Through.	On.						
mm.	mm.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.
.....	2.06	0 605	0.002	.....	.....	.....	.....
2.06	1.63	1.393	0.021	0.015	.....	.....	.....
1.63	1.44	0.448	0.022	0.016	.....	.....	.....
1.44	1.27	0.276	0.111	0.014	.....	0.002	.....
1.27	1.10	0.194	0.213	0.007	.....	0.002	.....
1.10	0.97	0.444	0.215	0.029	0.042	0.002	.....
0.97	0.84	0.813	0.422	0.070	0.010	0.005	0.002
0.84	0.68	0.487	0.367	0.116	0.007	0.005	0.001
0.68	0.57	0.271	0.351	0.172	0.016	0.005	0.003
0.57	0.45	0.146	1.779	0.676	0.067	0.013	0.010
0.45	0.36	0.034	1.191	0.420	0.158	0.032	0.028
0.36	0.28	0.018	0.515	0.843	0.591	0.111	0.090
0.28	0.24	0.005	0.047	0.574	1.056	0.426	0.088
0.24	0.20	0.007	0.054	0.668	1.771	1.959	0.433
0.20	0.15	0.007	0.012	0.120	0.973	1.950	0.572
0.15	0.12	0.002	0.006	0.028	0.380	1.792	0.717
0.12	0.10	0.005	0.002	0.009	0.063	1.028	0.206
0.10	0.08	0.009	0.004	0.009	0.061	0.972	0.694
0.08	0.00	0.014	0.002	0.004	0.018	0.346	0.569
Total.....	.....	5.178	5.336	3.790	5.213	8.650	3.413

Analyses of the above products giving percentages of copper.

On.	Run No. 11.	Run No. 12.	Run No. 13.	Run No. 14.	Run No. 15.	Run No. 16.
mm.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
2.06	8.74	8.62	.....	.....	.....	.....
1.63	8.71	6.72	2.06	.....	.....	.....
1.44	8.12	7.80	1.68	.....	.....	.....
1.27	5.12	8.00	2.53	.....	0.00	.....
1.10	1.48	8.16	3.37	.....	0.00	.....
0.97	0.57	7.81	5.62	0.16	0.00	.....
0.84	0.16	7.20	6.46	2.12	0.00	0.00
0.68	0.10	6.60	6.52	5.27	0.00	0.00
0.57	0.27	2.99	5.99	5.52	0.53	0.00
0.45	0.50	0.036	2.57	4.92	0.11	0.23
0.36	1.24	0.14	1.73	4.65	1.38	0.51
0.28	1.85	0.20	0.70	4.43	2.89	0.30
0.24	2.60	0.68	0.21	1.48	1.60	0.61
0.20	2.52	0.89	0.21	0.33	0.86	0.24
0.15	3.97	2.84	0.73	0.70	0.97	1.41
0.12	5.26	2.85	3.34	0.79	0.86	1.87
0.10	5.50	5.89	3.38	0.73	1.25	1.59
0.08	4.35	7.08	6.68	2.89	1.23	3.67
0.00	4.94	7.52	10.10	5.84	6.17	8.77

TABLE XIII.—*Results of Tests Nos.* 11 *to* 16 (*Tailings*).

Tailings sized and weighed. The weights computed in tons and fractions of a ton on the 100-ton basis.

On.	Run No. 11.	Run No. 12.	Run No. 13.	Run No. 14.	Run No. 15.	Run No. 16.
mm.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.
2.06	1.262	0.382	0.090	.....	.....	.....
1.63	18.750	8.348	0.988	0.012	.....	.....
1.44	11.199	9.814	3.081	0.109	.....	.....
1.27	12.096	20.091	7.017	0.213	.....	.....
1.10	4.368	10.091	4.732	0.467	.....	.....
0.96	2.688	7.747	8.303	0.893	0.029	0.149
0.84	1.878	11.943	11.283	2.067	0.017	0.015
0.68	0.557	9.258	9.154	2.819	0.033	0.020
0.57	0.151	5.693	9.412	5.457	0.141	0.030
0.45	0.045	4.481	22.336	24.613	3.718	0.587
0.36	0.011	0.945	9.221	13.126	4.870	2.392
0.28	0.006	0.209	4.419	17.356	16.205	16.015
0.24	0.004	0.028	0.896	9.864	18.463	10.332
0.20	0.007	0.020	0.345	8.467	17.150	19.193
0.15	0.008	0.024	0.128	2.493	12.623	11.521
0.12	0.006	0.013	0.040	0.705	6.076	12.217
0.10	0.004	0.006	0.017	0.150	1.152	5.838
0.08	0.014	0.017	0.022	0.122	1.923	5.579
0.00	0.022	0.038	0.030	0.044	0.481	2.093
Total.....	53.076	89.148	91.514	88.977	82.881	85.981

Analyses of the above products giving percentages of copper.

On.	Run No. 11.	Run No. 12.	Run No. 13.	Run No. 14.	Run No. 15.	Run No. 16.
mm.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
2.06	0.14	0.00	0.00	.....	.....	.....
1.63	0.03	0.01	0.00	0.00	.....	.....
1.44	0.00	0.01	0.00	0.00	.....	.....
1.27	0.00	0.00	0.00	0.00	.....	.....
1.10	0.00	0.00	0.00	0.00	.....	.....
0.96	0.00	0.01	0.00	0.00	0.00	0.00
0.84	0.00	0.00	0.00	0.00	0.00	0.00
0.68	0.00	0.02	0.00	0.00	0.00	0.00
0.57	0.00	0.01	0.03	0.00	0.00	0.00
0.45	0.24	0.07	0.03	0.00	0.00	0.00
0.36	0.45	0.16	0.02	0.00	0.02	0.00
0.28	0.92	0.16	0.04	0.01	0.09	0.02
0.24	1.01	0.23	0.05	0.02	0.00	0.00
0.20	0.62	0.60	0.04	0.01	0.00	0.00
0.15	1.33	0.41	0.15	0.05	0.00	0.00
0.12	1.38	0.19	0.43	0.00	0.00	0.00
0.10	5.08	3.05	1.07	0.03	0.00	0.04
0.08	5.98	4.00	0.52	0.21	0.04	0.05
0.00	5.67	1.63	1.56	1.82	0.24	1.13

A comparison of Tables VIII. and XIII. shows that the percentage of values increases in the fine sizes of the natural-feed tailings and in the classifier-feed tailings just as it did in the tests with galena, but since the natural-feed tailings have tons

of products, while the classifier tailings have only a few pounds, the classifier again is more satisfactory than the natural feed.

From these two tests on typical minerals, galena and cupriferous pyrite, we may generalize as follows:

An ideal sketch of the arrangement of the grains by a Wilfley table is given in Fig. 1, which is a reproduction of

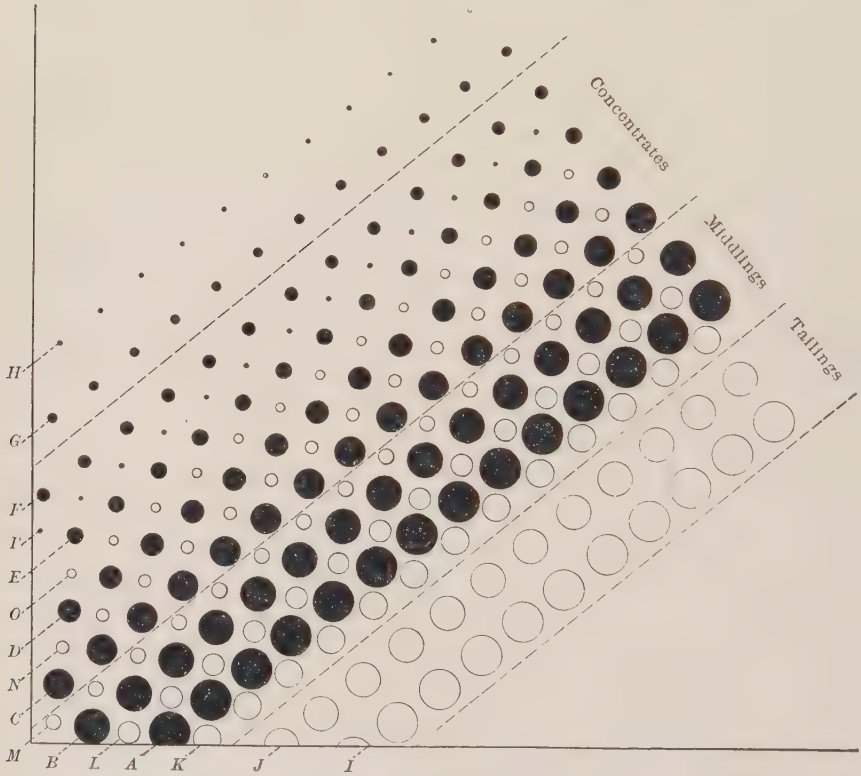


FIG. 1.—IDEAL SKETCH OF THE ARRANGEMENT OF GRAINS BY A WILFLEY TABLE.

Fig. 4 of my first paper.[3] This sketch shows that sizing by sieves would send together to the first table, grains, *I*, quartz, and heavy mineral, *A*; to the second table, grains, *J*, quartz, and heavy mineral, *B*, and so on. On the other hand, the perfect classifier would put together for the first table perhaps grains, *I*, quartz, and heavy mineral, *A*, *B*, *C*; for the second

[3] *Trans.*, xxxviii., 577 (1908).

table, grains, *J*, quartz, and heavy mineral, *D*, and so on. The first table of the classifier set will then have no more difficult separation to make than the first table of the sized set, because the grains, *I*, and *A*, have their natural tendency to be the same distance apart in both cases. On the other hand, on the finer tables we see that the sized feed puts together grains, *J*, quartz, and heavy mineral, *B*, while the classified feed puts together grains, *J*, quartz, and heavy mineral, *D*. The natural distance apart of the two minerals, then, is much greater in the case of the classified feed than in the case of the sized feed. There remains only to design classifiers sufficiently perfect to realize this natural advantage of classified feed over sized feed as applied to the Wilfley table.

The Future Gold-Output of Colombia.

BY HENRY G. GRANGER, CARTAGENA, COLOMBIA.

(New York Meeting, February, 1908.)

A RESIDENCE of 14 years in the Republic of Colombia, spent in almost continuous traveling and prospecting-trips, has given me an intimate knowledge of the resources of that wonderful country.

The man who best knows Colombia is unquestionably General Rafael Reyes, President of the Republic for the past three years, and with seven years more to serve in his present term. Such is the strength of his personality and the wisdom of his policies that his term of office will no doubt endure as long as he is willing to serve. The vastly improved economic situation and the brilliant outlook for Colombia are so directly due to the influence of President Reyes that a few lines concerning him will not be amiss, if they serve towards a better understanding of his personality. Born of a distinguished ancestry in the mountains of Boyaca, he is now 57 years old, just the age of his friend Porfirio Diaz when he began the work which has placed Mexico in the rank of great modern nations.

It was Reyes who came to Washington to protest against the action of the government of the United States of America in

reference to the establishment of the Republic of Panama, and, without entering into a discussion of this affair, it can be fairly said that there was not an American in Colombia whose feelings of justice were not outraged at the government action and the subsequent refusal to submit the matter to the impartial judgment of the Hague.

President Reyes is an accomplished linguist, speaking English, French, German, Portuguese, Spanish, and Italian. He has a remarkably powerful voice, which can readily be heard by a large assemblage. His height is about 6 ft., his weight exceeds 200 lb., and he has clear gray eyes.

Reyes was the leader of those who favored the approval of the Hay-Herran treaty, and to this fact he owes the Presidency, which he won by a narrow margin in a hard-fought contest from the late Joaquin F. Velez, the reactionary leader, whose influence caused the defeat of the treaty and the loss of the Isthmus.

President Reyes established a protective tariff which has resulted in the development of various manufacturing industries, while exports are constantly increasing. Any one desiring to establish an industry in Colombia can count on the cordial co-operation of the government. In former years an occasional source of revenue in the Choco was burning and pillaging the town of Quibdo, the commercial center of the principal placer-gold region of the country. Last spring an attempt of this kind was made; but the incendiary was caught before the flames had got beyond control. Telegraphic orders were sent from Bogota to the local authorities to declare martial law and try the culprit. The incendiary's plea of temporary emotional insanity was not accepted; and his ante-mortem statement warned all others against following his example.

The telegraph service, formerly execrable, has been brought at least to the Western Union standard of efficiency, and the rates are much lower. The number of public schools has trebled; and many manual-training schools, as well as a number of trade-schools, open at night for day-workers, have been established.

The agricultural resources of Colombia are as varied as can be conceived. The coasts and hot regions produce corn, sugar-cane, sea-island cotton, rubber, cacao, bananas, fibers, and all

the tropical fruits and vegetables; while the mountains and higher plateaus yield coffee, apples, peaches, potatoes, wheat, barley, and all the products of the temperate zone. The whole country is suitable for stock-raising, the pastures in large sections remaining green the year round. With a little care and attention mining operations can be conducted practically everywhere below the frost-line without the use of a can-opener.

The mining resources in this North Andean Republic are as vast and varied as the cordilleras which bear them. A map of the Department of Antioquia, showing the principal mining districts, is given in my paper on Mining Districts of Colombia,[1] and a sketch-map of the Choco region, showing the mining districts and the railroads built or under construction, in my paper, Gold-Dredging on the Choco Rivers,[2] to be presented at a later meeting. Coal is found nearly everywhere, lignite on the coasts and coking- and steam-coal in the interior. Iron-ores occur at many points, and are utilized commercially at Amaga in Antioquia and on the savannas of Bogota. Veins of copper-ore are encountered in Antioquia, Cauca, Tolima, Boyaca, and Cundinamarca. Many of these deposits are of high grade, but owing to lack of transportation-facilities they are as yet unworked, except in rare and insignificant instances. Emeralds occur in the mountains of Boyaca, and the famous Muzo mine makes Colombia the leading producer of this precious stone. This mine is government property, and the working is now being so systematized as greatly to increase the yield.

Springs where petroleum oozes out are of frequent occurrence. The oil has an asphalt base, and in general is similar to that of the Beaumont field. Rock-salt occurs in enormous masses near Bogota at Zipaquira, and in the hills bordering on the Upia river. With the extension of the railroads to the east of the capital, it is possible that Colombia may enter the nitrate market as a rival of Chile. At a high cost of transportation large shipments are being made from Santander of a high-grade asphalt of a quality used for japanning and lacquering.

Silver- and lead-ores occur in Antioquia, Tolima, and the Cauca. The most noted silver-mine is the Zancudo, at Titiribi, Antioquia, a controlling interest in which has recently been

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[1] *Trans.*, xxviii., 35 (1898).

[2] See later in this volume.

secured by French capital. Other noted silver-mines are the Santa Ana, in Tolima, under English control, and the Cerro de Pava at Supia, in the new State or Department of Caldas, which made the late J. M. Hernandez a millionaire.

The platinum-output of Colombia is second only to that of Russia. This precious metal, washed from the gravels of the Choco, is always found mixed with gold, sometimes one or the other metal greatly predominating. Although platinum occurs to some small extent in other parts of the Choco, its main sources are the Platina and Condoto rivers, with a few tributaries, and some of the streams running into the Cajon, all being tributary to the San Juan river. The metal is also obtained from the Agua Clara river and certain sections of the Bebera river, both of which flow into the Atrato, and from the Certegui, which is the main affluent to the Quito river.

On a recent trip to a California placer-deposit, I was told of the finding of platinum with the gold, and the manager of a certain property showed me with great pride a medicine-vial half filled with heavy metal which had been carefully collected in six months. It was about the quantity that a good diver on the Condoto river would have considered, with about 30 per cent. of gold intermixed, a fair afternoon's washing.

Though platinum occurs in the wide areas of the Choco in the stream-beds and the banks bordering on them, it has never been found in place. Don Francisco de B. Carrasco, the principal merchant of Istmina, owns the mine on the upper Condoto that has produced the largest nuggets of platinum, one of which weighed several pounds. The recent great increase in the value of platinum has stimulated the search for the source of the metal. A party under one of the sons of J. Henry White, the noted English engineer, is working systematically with this object in view.

Some engineers, who have made a careful study of the coal and iron resources of Colombia, predict her future greatness as a producer of the world's supply of fuel and steel. Many who see the abundant yields of her fertile soils care nothing for her mines, and say that with cheap transportation the agricultural exports of Colombia will rival those of Argentina in value if not in kind. But what made Colombia the famous Eldorado of the early Spanish adventurers was the bait that later drew

the British pirates to the bloody walls of Cartagena—gold. Before the coming of the Spaniard, this precious metal was used by the early Incas for making ordinary household utensils, drinking-cups, fish-hooks, and personal ornaments. Gold, pillaged or mined amid scenes that made the fiercest days of the "forty-niner" look like a Sunday-school, was sent home in shiploads. Spain was thus made a world power, and Colombia the foremost gold-producing country until the rush of "forty-niners" and the use of modern methods in California, South Africa, Australia, Russia, Canada, Mexico, and India, enabled these countries to pass her in the quantity annually produced.

With enslaved labor working at the muzzle of a blunderbuss, the Spaniard was exceedingly thorough in his methods. There is no use in attempting to work behind him in the stream-beds which he was able to bottom or divert, or the banks on which he could get water and dump to ground-sluice. There is nothing in Colombia to-day to attract the poor miner who expects to make an ounce a day with a rocker and pan. The Spaniard cleaned out all such places except in remote fastnesses where the Indian with his deadly blow-gun was supreme, and even at the present time these localities are beyond the reach of the ordinary prospector.

It is only natural that Colombian gold, which had enriched Spain, should be a popular theme in Castilian annals and in the diaries of her missionaries. But that all writers who have visited Colombia, from the famous Baron von Humboldt to the great Trautwine and Shunk, should dwell at length and repeatedly on the gold-mining resources is only comprehensible to those who can form some conception of the great ridge that, with its lofty back-bone of vein-studded porphyry, granite, and trap, separates the Choco and Antioquia, and the scarcely less rich sister range that separates the rivers Cauca and Magdalena.

In this great area of many thousand square miles it may be said that wherever there is gravel there is gold. If one were to average the known values of the places where the energetic Antioquian has wing-dammed the Porce or Nechi and so reached bed-rock, and the results obtained by strong-lunged divers going to bed-rock in periods of such drought as occur but once in a generation in the upper San Juan, Condoto, Tamana, Sipi, Andagueda, Atrato, Capa, Negua, Bebera,

Certegui, and Tumbutumbudo, and then average these phenomenal results with the washings obtained lower down on the same streams and along the bars and banks of the Cauca, Cabi, Cajon, Tanando, Murri, Quito, Icho, and a dozen others, a total would be obtained for the entire body of gravel that would make the brain whirl, and run into figures that would pass reason and comprehension. Yet the gold in these streams is but the comparatively insignificant scourings from the veins to be found everywhere in the cordilleras when the appropriate country-rock is reached. Many of these veins are being worked, but only such as will yield a return with the coarsest crushing through 11-mesh screens with crude, little, iron-shod wooden stamps wheezing up and down their irregular course, with no other mortar than that afforded by a heap of ore confined in a bin. The Zancudo, Frontino and Bolivia, and Marmato mines alone show evidence of modern methods, where machinery has been transported in sections over many leagues of precipitous mule-trails at enormous cost.

In this great range is located the famous Chaves mine, which made the late Bartolo Chaves the richest man in Colombia. The Institute has a sample that Manager Ricardo Eastman broke off from one of the last slabs chiseled out before water stopped the downward progress. Here, also, are the Rio Sucio mines, one of which was taken for a debt, and in a few days values were struck that produced a small fortune a day for a decade. Nearby are the mines of General Alfredo Vasquez Cobo, President Reyes's late Minister of Foreign Relations, who ever and again, as he sat at his desk, forgot canals and Brazilian boundaries and opened his drawer and gazed lovingly at pieces of ore, half of the weight of which is due to the multitude of specks of the yellow metal. He hoped for the appointment of his successor, so that he might devote himself to the equipment of his mills and make the fortune of his dreams.

Scores of small mills in Antioquia are scratching at ore-deposits that would supply modern mills. One great mountain-top of low-grade ore that would keep the Homestake's entire equipment running for centuries is being operated by 10 wooden stamps, whenever there is water enough going over the wheels, built on the ends of the cam logs, to run them all. Dozens of wide lodes that can be traced for miles are unworked because

they do not meet the Antioquian prospector's requirements of a thumb-nail covered with gold from a *batea*-full of the blossom rudely crushed between two stones and washed in a wooden pan. Hundreds of square miles of this mineral-belt have never been trod by a foot other than that of the Indian hunter. Ten years ago I predicted a quartz-development on the uninhabited western slope of this range, and later a portion of it was explored, and a small wooden mill was installed and is now running. A forty-eighth interest in the property has yielded $150 per month to General Benito Uribe, ex-Governor of Antioquia.

Recent gold discoveries near Neiva, on the upper Magdalena, on the eastern edge of the belt, have opened up a new section. Don Zoilo Cuellar, one of the most prominent gentlemen of Bogota, with the face of a *conquistador* and the manners of a damsel, generally carries in his hip-pocket one of the last batch of bars his son sends him from the monthly clean-up. Vicente Olarte Camacho, the eminent legal light, is hiding his books behind rows of samples from his mines. Pedro Nel Ospina, spoken of for President a few years ago, is ever extending his mining interests, and can always produce a sample or two from his pockets. Legislators from Nariño, the new State at the south of the Cauca bordering on the Ecuador line, bring to the Capitol Halls great blocks of gold-studded ore from the mines being opened at the sources of the gold that enriched the streams flowing to the Pacific.

In the light of the present and past, what of the future?

Much of the gold in the gravels of the river-beds will remain there, tantalizingly, until the end of time. Streams that cannot be diverted, and having huge boulders and hard bed-rock with a torrential current that makes dredging impossible, are hopeless as factors in the gold-supply of the world.

Some parts of the upper Magdalena and its tributaries from the west can be considered in the light of a dredging possibility. Various sections of the upper Cauca and some of its tributaries can unquestionably be dredged at a profit as soon as railroad-construction will enable gold-dredges to be taken to the ground. The ready field for gold-dredging, the preeminent field in the world to-day in the extent, accessibility, and workability and value of its gravels, is the Choco, or the region which is watered by the Atrato and San Juan rivers and

tributaries. The only drawback to this region, at present, is the lack of port-facilities at the mouths of these rivers to enable dredges to be built and fitted-out without excessive cost, or the risk of a sea-voyage from Cartagena or Panama, or direct from the United States.

Various attempts have been made in past years to dredge the gold- and platinum-gravels of the Choco river-beds, but none with digging- and gold-saving equipments such as modern practice has shown to be essential to successful work. Below Quibdo are the remains of two wrecked dredges, visible at low water, that were brought there so long ago that the oldest inhabitant can tell no more than that they were brought to dig gold. About 30 years ago another dredge was sunk a hundred miles below the gravel region, in transit up the Atrato, its unsheathed hull being honeycombed by the teredos that bored into it during its construction in the Gulf. A little later, still another dredge, rigged with a steam vacuum-chamber, was brought down. A few days' work sufficed to show the hopelessness of this system, nevertheless the sight of the gold inspired a further effort with centrifugal pumps, one after another of which was tried with different linings, but the constant clogging and breaking soon brought the inevitable shutdown of operations. Next, a floating steam-shovel was brought, which gave the best satisfaction as a digger, but the intermittent over-loading of the sluices made it impossible to save the gold despite the various alterations that were made in the hope of saving the abundance of gold found in the tailings. The last dredge taken to the Choco was, so far as digging and gold-saving is concerned, practically an exact duplicate of the famous Continental dredge of the Feather river, Cal., built at the same time by the same concern. It will be remembered that before the Continental dredge repaid the cost of cleaning up the sluice, its gold-saving devices had to be entirely rebuilt, and before it reached the present successful condition the whole elaborate multi-storied system of sluices and under-currents had to be rejected, together with the coarse revolving-screen and pump, and a New Zealand gold-table and stacker with fine screens installed. Though this Choco counterpart was operated under the disadvantage of insufficient capital to make the changes, and a lack of knowledge of what the changes should

be, and despite a multitude of mechanical defects that caused shut-downs three-fourths of the time, the clean-ups encouraged perseverance, until finally the multiplicity of broken buckets and break-downs caused the work to be abandoned.

However, it is stated that at last two modern dredges will be sent to replace these latest efforts during the coming year. If they reach the ground and are duly constructed and equipped, success is certain. There is nothing in the gold-dredging of the Choco that presents greater difficulties than the California or New Zealand fields, save in the upper San Juan, where the bed-rock is too hard to dig. But in this stretch the values, before reaching bed-rock, are so high as to insure a good profit even without scraping the bottom.

A profit has been made in New Zealand treating gravel of a value of 5 cents per cu. yd. and on the Feather river it has paid to treat 11-cent ground. In the Choco a safe average of value is said to be 30 cents per cu. yd., while there are many miles that contain 60 cents and more. In many sections the gravel spreads into flats of greater or less width along the banks, which eventually will be dredged, but for years to come the work will probably be confined to the river-bottoms, where there is less barren over-burden to remove.

In 1908 there was shipped to the Choco a dredge which is claimed to embody all the necessary features for successful work. If the company sending it is not cramped for capital, and can get the dredge erected and on the ground, in good condition, it should make very satisfactory returns, although perhaps far short of the fabulous results set forth in its prospectus.

As soon as a properly-designed dredge is put in operation on a Choco river, the future of gold-dredging in that region will be solved, and bearing in mind the hundreds of gold-dredges operating in California and New Zealand, and also the fact that the Choco placer region has twice as much territory as both the regions in California and New Zealand together, an idea may be had of the potential importance of this branch of the gold industry in Colombia.

The construction of the Colombia Central railroad, the only standard-gauge road in Colombia, is to be continued with American capital. The northern terminal is in Colombia bay, Gulf of

Uraba, opposite the mouth of the Atrato. In the near future, this railroad will afford facilities for sending machinery and dredge-hulls to the Choco much cheaper than at present, and with no risk. The proper development of the San Juan dredging-fields will require adequate facilities for unloading machinery and for constructing dredges at the mouth of the Atrato, and an arrangement with the government for the establishment of a port at that place.

The development of the quartz-mining belt, previously described, depends solely on transportation. It is only with considerable difficulty and expense that machinery can be delivered to the new Neiva field by the various steamer- and rail-connections along the Magdalena river, involving a special charter up the river south from Girardot. The extension of the Dorada railroad and the little Tolima railroad will facilitate transportation to some of the mines of the eastern slope of the gold-belt. The construction of the Colombia Central railroad will provide direct transportation from the coast through the very heart of the belt, and this will mean the equipment with modern steel mills of all the mines now running with small wooden mills; also the development of many mines that will not pay under the present crude milling-methods. Moreover, it must be borne in mind that, excepting a few mills, such as at the Zancudo mines, all the mills running in Colombia to-day are operated by simple over-shot water-wheels. There are known lodes at a distance over rocky cliffs too great to bring water enough to run a mill, that can later be reached by electricity generated at any of the many rapid streams flowing unchecked into the Cauca.

With the opening-up by rail of this vast treasure-house, it is entirely reasonable to expect that each light wooden stamp will be replaced by a heavy steel one having a twelve-fold increase in crushing-capacity, and a three-fold increase in recovery of gold owing to the finer crushing. It is also evident that the number of mines now being worked will be at least doubled. The future for gold-mining in Colombia depends in no degree on startling discoveries, but on the systematic development of the known resources. The return of Colombia to former leadership among gold-producing countries is simply a matter of the development of gold-dredging in the Choco and

the installation of milling-plants attendant on rail-transportation through the greatest known gold-ore bodies on earth.

Those who have studied the great resources of Colombia believe that under the able government of President Reyes the country has entered on an era of progress unsurpassed in history. Charles M. Pepper, the distinguished American traveler and journalist, now touring the Southern Continent for the Department of Commerce and Labor, recently remarked to me in Bogota: "I know of no Latin-American country whose prospects of peace and prosperity are better than those of Colombia."

It was my original intention to extend the present paper so as to bring the history of the mining industry of Colombia fully up to date, but after a careful study of all the data now available, I decided to postpone this work until later. In the near future I expect to make an extended trip through the principal mining districts of Colombia, during which I shall make careful notes and observations, so that on my return I may be able to prepare another paper that will more faithfully represent the existing conditions and future prospects of the mining industry of the country.

Dip and Pitch.

BY R. W. RAYMOND, NEW YORK, N. Y.

(New York Meeting, February, 1908.)

Prof. Henry Louis, of Armstrong College, Newcastle-on-Tyne, England, a distinguished member of this Institute and other technical societies, has recently sent to the Institution of Mining Engineers, and to the Institution of Mining and Metallurgy, in Great Britain, a communication, the substance of which is as follows:

Professor Louis calls attention to the absence of any recognized English term for the definite description of a frequent phenomenon—namely, the oblique position of an ore-body in the plane in which it lies. In this connection, Professor Louis says that in America the term "pitch" has been occasionally applied to this relation, and proposes that, in the literature of ore-deposits, it shall be hereafter restricted to this particular meaning, the angle of the pitch, like that of the dip, being always measured from the horizontal, and the dip being always taken at right angles to the strike, while the pitch is taken on the plane of the dip, in the direction of the strike. Thus, for example, in a vein or bed striking N–S., and dipping, say, 45° E., there might be an ore-body, the axis of which pitched northward, at an angle of 45° below the horizontal line, as measured on the plane of the inclosing vein or bed. If the term "pitch" had this universally recognized meaning, the situation could be simply and perfectly expressed by its use, in connection with the general data of strike and dip. As Professor Louis says:

> "This suggestion has a certain practical value, and is not merely academic. Every mining engineer of practical experience in such types of ore-deposit will have met with cases, in which a vertical shaft has been sunk with the object of cutting the deposit, but has missed it, because the dip alone has been taken into account, while the pitch has been overlooked."

He adds the acute observation that such an oversight is much likelier to occur with regard to a condition not definitely desig-

nated, and therefore not invariably recorded. Finally, he points out that if the dip and strike of the inclosing vein, and the pitch of an included ore-body, be stated, the actual dip and strike of the latter can be determined by a simple geometrical construction, and hence need not be directly measured. Of course, if a given ore-body pitches 45° N. in a bed or vein which strikes N–S., and dips 45° E., that ore-body must be constantly making to the eastward with increasing depth; but the direction which it thus pursues can be, as Professor Louis shows, easily determined without additional surveys, from the dip and strike of the inclosing zone and the pitch and general direction of the body itself.

To the proposal thus advanced by Professor Louis, I would give my hearty assent, pointing out, however, that the American usage which embodies it, and which he describes as "occasional," is practically universal. I know, at least, that I have followed it for 40 years; and I do not think any instance to the contrary can be found in the 27 volumes of our *Transactions* which I have edited.

On the other hand, it is always somewhat perilous to limit the meaning of a term, previously employed in a looser sense; and there is no doubt that "pitch" has been often, and is still sometimes, used as synonymous with "dip." In view of that fact, I would suggest that, while mining engineers, following Professor Louis's excellent suggestion and our general American practice, use the term "pitch" exclusively in the sense he proposes, they take pains, at the same time, to avoid misunderstanding by invariably stating, together with the "pitch" of a body, the dip and strike of the vein or bed in which it occurs. Indeed, without these explanatory particulars, a statement of the "pitch" would have relatively small value; whereas, taken together with them, it would not only preclude all misunderstanding, but also furnish to the mining engineer all the data required for the planning of underground work.

Effect of Humidity on Mine-Explosions.

BY CARL SCHOLZ, CHICAGO, ILL.

(Chattanooga Meeting, October, 1908.)

During November and December, 1907, four serious mine-explosions occurred in the Appalachian coal-field, which resulted in the loss of nearly a thousand lives and caused an enormous damage to property. These accidents attracted more than the usual attention; coal-trade and engineering papers contained many comments and suggestions on the cause and prevention of accidents; the daily press in many instances condemned the coal-operators and charged these accidents to their negligence, or their anxiety to make money without adopting precautionary measures. Immediately after each accident the respective State authorities ordered close investigations to be made, with the view of establishing the cause and suggesting remedies to prevent recurrence. Representatives of the Technologic Branch of the U. S. Geological Survey, to whom the investigation of mine-accidents had recently been delegated by the Secretary of the Interior, visited the mines and co-operated with the local authorities in their determinations. It may be safe to say here that an investigation of a mine after an explosion has occurred discloses but little, because the causes have been removed and conditions have been entirely changed.

On Jan. 8, 1908, the coal-operators of West Virginia organized an association at Washington for the purpose of making extensive investigations, and sufficient funds were pledged to carry out this work. At this organization meeting, the representatives of the Geological Survey present had no suggestions to offer as to the cause of these accidents.

An examination of the various reports concerning these accidents indicates that the explosions were caused by the ignition of gas or dust; they show the point of origin and the direction in which the force was expended, and give a detailed description of the damage done. Very little has been said as to what

should be done to prevent these explosions, and the object of this paper is to give the result of several years' observation of conditions which I have noticed during 15 years' connection with coal-mines in various fields of the United States.

The striking features developed by these investigations are:

1. Explosions occur more frequently in the colder months of the year; the colder the winter the more frequent the explosions. If a certain district has extremely cold weather and other sections of the country are comparatively warm, the latter sections are freer from explosions.

2. Mining-fields located in higher altitudes are more productive of explosions than those at lower elevations.

3. The hygrometric condition of the atmosphere has the greatest effect upon the cause of explosions.

Every practical mining-man knows that the majority of explosions take place between Nov. 1 and Mar. 15. It is well to say here that there are many explosions of a minor character which result in no loss of life, or perhaps one single death only, of which but little is published in the daily or technical press. Every mine-examiner instinctively feels danger when he enters the mine on a cold, crisp morning.

The fact that altitude and general climatic conditions enter into mine-explosions can very readily be verified by an examination of the weather reports showing the general climatic conditions which existed in the various localities when accidents have happened.

The principal fields of mine-explosions in the bituminous districts are Pennsylvania, the eastern portion of West Virginia, Alabama, Oklahoma, and Colorado; and since coals produced in these fields are high in volatile matter and low in moisture, it is very clear that the chemical composition of the coal has much to do with the generation of mine-gases and coal-dust.

In examining the records of explosions in West Virginia, it will be noted that most of them occur during unusually cold weather, and but few in the warm season when the trees are in full foliage. In Oklahoma the record indicates that the greatest number of explosions have followed an unusually dry season. It should be noted that this portion of the country is not densely timbered, and contains but limited bodies of flowing

water. The same condition applies to Colorado to an even greater extent.

The two conditions above mentioned, when taken in connection with the visible results which they create in the mine by the deposit of excess of moisture during the wet season, or when humidity is high, and the creation of dust during the winter months or the very dry season, undoubtedly lead us to believe that the hygrometric condition of the ventilating-currents has much to do with mine-explosions; and since this is a matter which can be controlled without much expense, it is believed that the application of a vaporizing-system in mines which are dusty and generate fire-damp will prove, at least in part, a remedy.

After examining many text-books and reports, I noticed that very little reference is made to the hygrometric condition of the atmosphere in connection with mine-explosions. The only reference is given by William Tate.[1] Some of the more recent publications refer to the advisability of ascertaining the moisture in the air, but do not state the effect which humidity has, and how the required degree of saturation should be brought about. That humidity in a mine-ventilating current is desirable and necessary, is generally understood, and is emphasized by the recommendation usually made by mine-inspectors that roadways be sprinkled with water in order to settle the dust. The adoption of appliances to settle the dust will prove the best safeguard against mine-explosions.

One of the most striking remarks on this subject was made by Samuel Dixon before the meeting at Washington above mentioned. He said that mine-explosions began to occur as soon as improved ventilating-fans of high efficiency were adopted, and that as long as the mines were ventilated by furnaces and fire-baskets, mine-explosions were practically an unknown condition. This statement is true in so far as in summer very little water was carried into the mine, and in winter very little was taken out, as is shown by the following statements.

The mines of Oklahoma offer special facilities for the observation of the effect of climatic and hygrometric conditions upon explosions, because the coal is high in hydrocarbon and low in moisture; the outside temperature ranges from 90° F. in the

[1] *Questions and Answers for American Mine Examinations*, p. 27.

summer to 10° F. in the winter, and the hygrometric condition has a wide range, because of the excessive rain in the summer, which is followed by a prolonged dry period in the fall and winter. The observations inside the mines are facilitated by the noticeable effect which humidity has upon the roof during the wet season and the number of explosions which usually occur during the cold, dry season. I therefore selected the mines in this field for a series of observations which were carried on during the past 18 months.

The first purpose of these investigations was to stop the slacking of the roof, which occurs during the "sweaty" season, beginning about May 1 and lasting until the middle of July. During this period a heavy deposit of moisture on the roof causes the slate to slack, especially on the in-take air-ways and near the place of in-take. On account of the high cost of timber, this condition considerably increases the production-cost. From August to November the mines become very dry, and are very dusty for the next three or four months, during which time explosions occur. One fortunate condition in this field is the fact that the veins are pitching, and the water usually runs along the entries, although the beneficial effect of this condition is not generally understood or appreciated.

Observations indicate that when the outside temperature ranges from 75° to 90° F., the inside temperature fluctuates from 72° to 76° F. In the winter, when the outside temperature ranges from 30° to 55° F., the inside temperature ranges from 60° to 64° F. In the summer, when the warm-air current enters the mine, the temperature falls upon coming in contact with the cold ribs along the entries, and the excess vapor is deposited as sweat on the roof and the roadways, so that no dust exists. The mine, of course, becomes drier nearer the up-cast, since a part of the humidity is absorbed by the freshly-mined coal. In the winter, when the inside temperature is higher than that of the air outside, the cold ventilating-current, upon warming and expanding, absorbs all the moisture available. If there is no water available, the relative humidity in the air-current is very low. For the purpose of illustrating this more clearly, the following data, representing actual readings, will be of interest:

With an outside temperature of 80° F. and relative humidity

of 75 per cent., at a barometric pressure of 29.2 in., a ventilating-current of 75,000 cu. ft. per min. carries into a mine, invisibly suspended in the air, during a period of 24 hr., 15,200 gal. of water. Upon cooling to a temperature of 75° F., not only would the mine-current be completely saturated, but there would be deposited in the mine nearly 1,000 gal. of water per day. In the winter, however, with a temperature of 32° F. and a relative humidity of 95 per cent., upon entering the mine and warming to 62° F., the relative humidity of the ventilating-current is diminished to 33 per cent., unless an opportunity is given for the air-current to absorb more moisture from running water or other sources. In order to saturate completely this warmed air-current of 75,000 cu. ft. per min., about 9,000 gal. of water per day is required. This is the reason why dust is generated in the winter in the better-ventilated mines, as stated by Mr. Dixon, and the assumption is well borne out by the accident at the Monongah mines, which had a ventilating efficiency of the highest order. The more cold air forced through a mine in the winter, the drier will that mine become; the more air forced into a mine in the summer, the more moisture will be deposited along the in-take entries, and where the roof is slate, the more difficulty will be experienced in keeping up the top. It is also well known that humidity in the air has an effect upon the ignition of gases, because the fine particles of water invisibly suspended in the atmosphere absorb much of the heat in combustion. For comparison, it may be said that an air-current of a temperature of 62° F., fully saturated with moisture, requires approximately 7 per cent. more heat-units (B.t.u.) than dry air to reach a temperature of 1,213° F., which is the point of ignition of fire-damp.

A water-spraying system, consisting of a number of small sprays distributed over the first half of the distance which the air-current traverses, will have the effect of preventing the formation of dust and moistening the gaseous mixture. The sprays should be placed near the roof, and discharge the water in the direction of the air-current. This arrangement will have an additional beneficial effect in mines generating much fire-damp, the descending water-spray serving to break up any stratification of gases that may exist in the entries. The proper quantity of water to be vaporized will depend upon altitude,

climatic conditions and character of the coal. The only danger to be guarded against is the use of an excessive amount of water, which would result in cutting a slate roof; if coal or sand-rock prevails, no limit need be established, because, with the lower temperature in the mines in this country, the danger which was pointed out by the British Colliery Commission, that excessive humidity injuriously affects the health of the miner, does not exist, the temperature of the English mines being from 90° to 95° F.

A large number of readings taken throughout the various seasons of the year in the Oklahoma mines, indicate that in the summer at the up-cast the relative humidity rarely falls below 75 per cent. Theoretically, the air-current should be fully saturated; the discrepancy can only be explained by the statement that the freshly-mined coal and the dust incident to mining absorb the difference. In winter, the drop in the relative humidity in a dry mine is very striking; and attention is drawn to the fact that in mines where shot-firers are employed, when explosions occur, they usually happen directly after the first few shots are set off, which must be due to the condition that the relative humidity at the point of up-cast is much lower, because what little water was contained in the down-cast was absorbed by the workings through which the ventilating-current first traveled.

A series of analyses of air were also made, and samples of air were taken in the mine at the end of a day's work, and again from the same point after the mine had been standing idle for 48 hr. The ventilating-current was maintained at a uniform rate, and there was no appreciable change in the atmospheric pressure. The first analysis showed that 0.05 per cent. of methane existed in the air-current next to the floor, which increased to 1.5 per cent. near the roof. The explanation of this condition is that the travel of trips and men through the mine-workings, and the fluctuation of the ventilating-pressure, due to the opening and closing of trap-doors, thoroughly mixed the air-current and prevented the stratification of the gases. The second analysis showed no methane near the floor, but 5 per cent. near the roof, which indicates that the comparative quiet and smooth travel of the ventilating-current during the idle period aided the stratification of the gases. This theory is borne out

by readings with a self-recording pressure-gauge, which showed many violent fluctuations during the day, and produced a practically straight line during the night.

The advantage of a spray, therefore, is that the vapor will break up stratification and mix the gases over the entire cross-section of the entry. This action can be repeated as often as necessary by the spacing of the sprays, and the requirements can easily be determined by hygrometer-readings and gas-tests. There would be no advantage in or necessity of operating these sprays during the summer, unless indicated by hygrometer-readings in very dry mines. The sprays used in conducting these experiments, and now used in several of the mines under my management, are manufactured by Paul Lechler, Stuttgart, Germany; but there are several manufacturers in the United States who produce similar apparatus which can be adapted to suit the conditions, as, for instance, the American Moistening Co., of Boston.

The principal benefit derived from sprinkling water by means of water-boxes or hose-lines, as now generally practiced and recommended, is that moisture is provided for absorption by the dry mine-air. As far as the settling of dust is concerned, unless the sprinkling is very thorough and often repeated, the results are not satisfactory, because a deposit of coal-dust is impervious to water unless thoroughly mixed with it; a thin moist coat will form on the surface, beneath which the dust is as dry as ever. The cost of labor in this method of sprinkling is very high and the service performed is spasmodic, and unless constantly looked after by foremen or superintendents it is likely to be neglected. The advantage of a spraying-system is that, in addition to preventing the formation of dust, the sprays can be utilized to prevent stratification of gases at night or on idle days, and this advantage can be obtained at a cost of installation but little greater than that of the hose system, as it involves only the addition of spray-boxes at a cost of $2 each. There is no expense connected with this, because most mines employ pumpers on idle days and at nights, and the shaft pressure can be used in the sprays; or if this is not convenient, in most places some outside supply can be connected which will insure continuous operation. It is believed that the operation of such a spraying-system, by reason of the continuous and

automatic protection which it furnishes to the mines, is the most economical, and at the same time the most important step which coal-operators can take to safeguard their mines.

I recommend that mining engineers and coal-operators, in fields subject to gas-explosions, obtain, for their own information, a record for a year, taken once a week, both outside and at the point of up-cast, to show the temperature and relative humidity, and from these results compute the amount of water which is carried into and taken out of the mine. It is predicted that the results will be astonishing to a great many.

Many conditions exist showing the effect of relative humidity upon everyday life, which were not comprehended for a long time. As an illustration, the textile-manufacturing business was very unsatisfactorily conducted in the United States compared with England, and it was only with the adoption of air-moistening apparatus that the manufacturers were able to improve the product, the relative humidity in England being so much higher by reason of the proximity to large bodies of water and natural damp conditions. Again, the introduction of breakfast foods in England by American manufacturers proved an absolute failure, because of the rapid absorption of water, which resulted in mold.

Likewise, it is safe to believe that if English mines could be transferred to Oklahoma or Colorado, they would become very unsafe by reason of the change in the climatic conditions; and, *vice versâ*, if we could move our Oklahoma mines to England they would be very safe. Gas rises more rapidly in an atmosphere free from humidity, as is demonstrated by the ascent of chimney smoke on a cold, crisp morning; and conversely, the opposite condition prevails in foggy weather. Likewise do gases rise to the roof of the mine and accumulate there in dry atmosphere; if the air is moist the ascent will be much slower.

Another comparison between the conditions in a mine and dry and wet weather may be made by the operation of internal-combustion engines. The consumption of gasoline in the summer is much greater than in the winter in the same climate; in the warm weather the excess of humidity present in the mixture absorbs a certain amount of heat-units before the vapor develops effective power; consequently, an automobile consumes more fuel in the summer than it does in the

winter under the same conditions. A gaseous mixture containing dry air becomes explosive with a much smaller percentage of methane, and the higher the ratio of humidity in the mine and the ventilating-current the less is the danger.

I respectfully invite comment and criticism on the opinions set forth in this paper, and I will gladly answer in detail any discussion by members of this Institute.

The Hardinge Conical Pebble-Mill.

BY H. W. HARDINGE, NEW YORK, N. Y.

(Chattanooga Meeting, October, 1908.)

DISREGARDING for the present the economic side of a new device, let us turn to that feature of the conical pebble-mill which is of interest from a scientific point of view, and consider the device as a puzzle in which the theories of the cylindrical type of mill have been increased.

Many articles have been written on the theory of the tube-mill, and explanations have been made which do not explain; but these have evolved suggestions of practice which have resulted in changes in the tube as to length, diameter, feed, discharge, speed of rotation, rotation on hollow trunnions, rotation on rings, and changes in the operation as to quantity and size of pebbles, charge of load above or below the axis, crushing by causing the larger pieces of the material of the charge to act as grinders, whether crushing is abrasive or impactive, etc.

The practical worker in crushing is first interested in the endeavor to reach an economic means of accomplishing a desired end; but the improvements in crushing-practice have been largely left to the manufacturer of machinery, who, as a rule, has little, if any, knowledge of the specific requirements—such as granulation, sizing for concentration without pulverizing the metals, instead of pulverizing the rock or gangue; the metals being the more friable, while the rock is tough and not so easily disintegrated. A reversal of this practice is the aim of sliming direct for cyanidation or pulverizing for cement, etc.

In the ordinary cylindrical mill, having a diameter of 4 or

5 ft. and a length of from 15 to 27 ft., it has been found that the maximum size of the particles of the charge must not exceed 1 mm. (about 25 mesh). Some of the charge, upon entering the mill, is immediately reduced to a slime. The work upon the individual particle having been finished, it should be passed out of the mill; but, instead of this, the finely comminuted material is retained in the tube until it works its way through 20 ft. or more of the mill, interfering with subsequent work, and absorbing energy by acting as a springy compressible medium between the pebbles.

In ordinary practice, both ends of the cylinder or tube are loaded with the same quantity and sizes of pebbles, and when these are reduced in size through gradual wear, the larger pebbles expend a further amount of power in acting upon the smaller ones, as well as in regrinding particles already sufficiently reduced (or which, if they are to be further reduced, should be ground by the smaller pebbles. For the much advocated "angle of nip" applies to a pebble as well as to a roll: and, if the action within the tube-mill is not one of rotative crushing, this practice at least uses the same medium to crush the 0.5-mm. and the 1-mm. size—a system as wasteful of power as using a sledge-hammer to drive tacks as well as spikes. The work is performed in both cases, but in the former it is accomplished only at a great waste of energy, as well as a nullification of the desired effects, by carrying the work to an unnecessary or an undesirable point.

The tube-mill of the past few years, like the action of the sledge-hammer upon the tack, has been accepted because it has filled a long-felt want, and has performed work more efficiently than the still more energy-wasting stamp for fine grinding. It has, however, not only done its work, but over-done it, by using a 3-in. pebble to crush 25-mesh sand-particles, having a relation of mass of 1 : 238,000, and then using the same power and crushing-medium to crush 60-mesh material, in which the relation is 1 : 2,700,000, or more than ten times that of the first case.

In order to correct this enormous waste of energy, as well as to approach more nearly the desired results, a pebble-mill of odd form has been designed which utilizes principles of physics to attain maximum results for the energy expended, and control

size without regard to the gravity of the particle. While it cannot be said that this form of mill has carried these results to a finality, a great stride has been made in that direction, as is shown by a study of Fig. 1, which presents the comparative peripheral speeds per minute at different sections of the mill, and a diagrammatic comparison of the size and weight of the crushing-pebbles and the particles to be crushed. The great increase in the difference as the sizes decrease is comparatively equalized by the diminished fall and reduced peripheral speed.

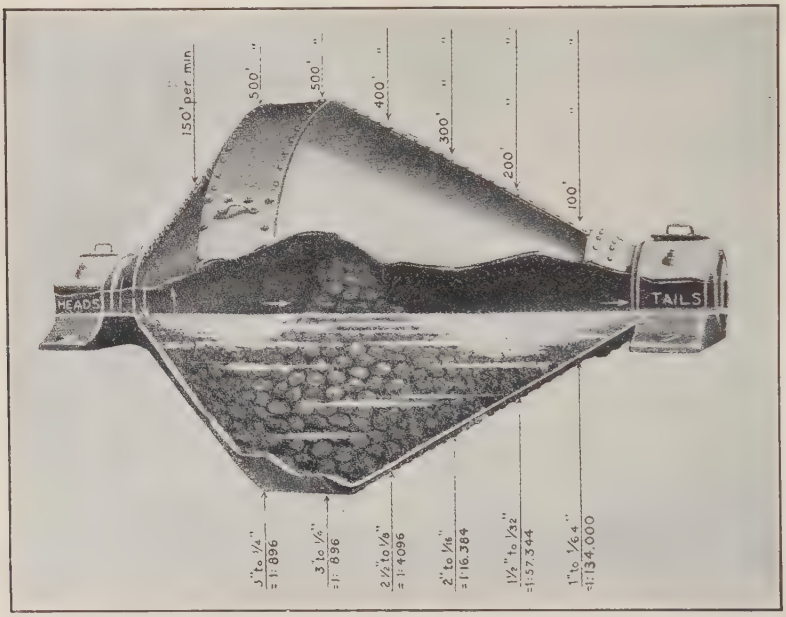

FIG. 1.—THE HARDINGE MILL, SHOWING COMPARATIVE PERIPHERAL SPEED PER MINUTE AT DIFFERENT SECTIONS, AND COMPARATIVE SIZE AND WEIGHT OF CRUSHING-PEBBLE AND PARTICLE TO BE CRUSHED.

The discovery of this special shape of mill arose from experiments necessitated by excessive cost of crushing. I refer to the gradual sizing-effect, not only of the material crushed, but of the crushing-bodies as well, for it is found that, aside from the crushing caused by the greater fall of the pebbles of larger diameter in the zone of greatest speed, the next smaller pebbles, through a sizing-action, due to two or three different causes, take a place up the incline of the outlet end of the mill, and act upon relatively smaller sizes of particles to be

crushed. While this sizing is apparent to the eye, the chief cause resolves itself into a scientific puzzle. My engineering friends have attributed this sizing-action to no less than ten different causes. In one instance only has the main or fundamental principle been explained.

If any one takes an interest in this subject, from either an economic or a scientific standpoint, let him take two ordinary glass laboratory-funnels, from 5 to 7 in. in diameter, preferably ribbed on the inside; place them base to base, first partly filling one with dry sand and pebbles, ranging in size from fine particles to coarse gravel; hold these funnels together with adhesive tape or paper labels; revolve at a slight angle from the horizontal, and observe the results. An opinion will doubtless be formed immediately as to the cause of the unexpected results, but, upon further observation, this opinion will probably be changed two or three times before a satisfactory conclusion is reached.

The following comparison of actual results, obtained by crushing equal quantities of material in a conical mill and in an ordinary tube-mill, is of interest. The charge for the conical mill was of 4-mesh size, while that for the tube-mill had already been crushed and passed through a 25-mesh screen. A 0.25-in. grain contains more than 200 particles of 25-mesh size.

		On 40-Mesh Screen.	On 60-Mesh Screen.	On 80-Mesh Screen.	On 100-Mesh Screen.	Thro' 100-Mesh Screen.
		Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.
Conical mill; with 2,000 lb. pebbles; consuming 15 h.p.	Charge.	75.5	6.6	5.5	2.3	9.5
	Product.	0.0	1	5.4	2.5	90.8
Tube-mill; with 11,000 lb. pebbles; consuming 48 h.p.	Charge.	1.5	13.5	17.0	17.0	53.0
	Product.	0.0	0.5	1.5	0.2	96.5

A study of Fig. 1 will help to explain the astonishing results given by the conical mill.

The total weight of a conical mill (except the lining), having a capacity of from 2 to 4 tons of material per hour, is 8,500 lb., and the floor-space occupied is 7 by 10 ft. The charge of

pebbles weighs about 2,000 lb., and the power required for operation is from 12 to 15 h.p. These mills can be arranged singly, in series or in tandem, and the large amount of aëration which takes place during the crushing suggests promising results in cyaniding ores during crushing, which, if attained, will greatly simplify present milling-practice.

A full-sized conical mill has recently been erected near New York City for the purpose of conducting crushing-tests on a large scale, and with certain ores the results have far exceeded the most sanguine expectations. A test on magnetite ore from Pennsylvania gave the following:

	On 0.25-in. Screen. Per Cent.	On 20-Mesh Screen. Per Cent.	On 40-Mesh Screen. Per Cent.	On 60-Mesh Screen. Per Cent.	On 80-Mesh Screen. Per Cent.	Through 80-Mesh Screen. Per Cent.
Size of ore fed, . .	53.2	30	2.2	2.2	1.6	10.9
Size of product, . .	0.0	0.3	0.6	1.2	2.5	95.0

The ore fed to the mill was "run of mine," which was first shoveled by a coke-fork having tines spaced 1 in. apart, the coarse material being rejected, so that the undersize was 1 in. and finer. The mill contained 2,000 lb. of 2- and 3-in. pebbles. The ore was crushed to 80-mesh size at the rate of 4 tons per hour at an expenditure of 17 h.p. This type of mill appears to be specially adapted to the use of lumps of ore, instead of the usual foreign bodies, for grinding. In a separate test on Pennsylvania magnetite, pieces as large as 8 in. in diameter were fed to the mill and the charge was ground at the rate of about 2 tons per hour, the product being reduced to a fineness corresponding to the results obtained in the first test on magnetite mentioned above.

Whether a mill of this character, or, in fact, any fine grinder, can be adjusted for fine or for coarse grinding appears to be an open question, but the following data, obtained in actual practice, show that the conical mill possesses this qualification.

For fine grinding, a 6-ft. conical mill recently installed at the plant of the Enterprise Mining Co., Cooney, N. M., for fine-grinding gold-ores prior to treatment by the cyanide process, is reported by C. F. Leng, President of the company, to be very satisfactory, as to both capacity and small amount of power required.[1]

[1] Private communication to the author, Feb. 26, 1908.

For comparatively coarse crushing, the conical mill in operation at the sapphire-mines of the American Sapphire Co., in Yogo county, Mont., is reported by John T. Morrow,[2] the President of the company, to give excellent results. This mill has a crushing capacity of from 5 to 8 tons of material per hour. The interior surfaces of the cones are covered with a series of angle-irons placed longitudinally, and soon after the charge is made and rotation begun, the spaces between these irons become filled with pieces of the material to be crushed, thus forming a protective lining for the mill. A special feature of the operation is the control of the size of the crushed material passing out from the mill, which can be made to vary from the finest slime to pieces $\frac{5}{8}$ in. in diameter, with the exception of the fines naturally produced in all divisions of larger to smaller sizes.

Requirements of a Breathing-Apparatus for Use in Mines.

BY WALTER E. MINGRAMM, NEW YORK, N. Y.

(Chattanooga Meeting, October, 1908.)

THE construction of rescue-apparatus on the principle of furnishing the wearer with air from a tank containing it under high pressure was given up by inventors about 20 years ago. Such an apparatus must supply continuously, for not less than two hours, nearly 50 liters of pure air at atmospheric pressure per minute, or a total of 6,000 liters (210 cu. ft.); and a cylinder containing this amount of air, under 2,000 lb. pressure, would weigh 180 lb. or more.

Although apparatus has been recently constructed utilizing liquefied air, no experiments have been made to determine the regularity with which the air is generated, nor has its chemical composition been determined. Moreover, the chief objection to such an apparatus is the difficulty and cost of keeping on hand a large quantity of liquefied air.

The system of furnishing the wearer of a breathing-apparatus with air which has given the best results, and has found

[2] Private communication to the author, May 28, 1908.

employment in all the devices now in practical use, is that in which the air contained in the human lungs at the beginning of the work is regenerated by the apparatus. By means of chemicals, such as caustic potash and caustic soda, the carbonic acid gas, CO_2, exhaled is absorbed, and the oxygen that has been consumed is replaced from a tank containing this gas under high pressure. The pressure of the oxygen (which is 1,875 lb. in the two cylinders at the beginning of the work) is automatically reduced to 75 lb., and is sufficient to keep the air inside of the apparatus in constant circulation, and to force it to pass through the chemicals provided for the absorption of the CO_2 gas.

In the attempt to construct an apparatus which would (1) furnish the wearer with a sufficient supply of air; (2) replace the amount of oxygen used up by the lungs during the time of two hours, and (3) thoroughly absorb the carbonic acid gas exhaled during the same period, a number of interesting experiments have been made by Bernhard Draeger, the inventor of the life-saving apparatus which bears his name.

Concerning the amount of air which a person requires during hard work, few facts were known until very recently. Generally, it was supposed not to exceed from 16 to 20 liters per min. Experiments made with three different individuals (the exhaled air being measured by a so-called spirometer or gasometer) have given the following average figures:

	Quantity of Air Required for 1 Min. Liters.
A person sitting down,	8.5
A person after a walk of 1,000 ft.,	11.5
A person after marching over 2,000 ft.,	16
A person lifting heavy weights,	35
Two persons, in a stooping position, carrying a third person,	30
A person racing over a distance of 900 ft.,	50

On the basis of these figures, a rescue-apparatus which will enable its wearer to do any kind of hard work must furnish not less than 50 liters per min. during the entire working-time of two hours. The Draeger apparatus meets this requirement.

During heavy work, and particularly after, there is always a period of a few seconds during which the amount of air required is momentarily increased to even more than 100 liters per min. A life-saving apparatus can only secure absolute

safety if it assists the workman unconsciously and without becoming oppressive during even the heaviest breathing. This requirement, also, is met in the construction of the Draeger apparatus.

It is therefore necessary to know the quantity of oxygen that must be supplied by a rescue-apparatus. The amount of oxygen used up by the lungs during rest is ascertained to be about 0.3 liter, gradually increasing to 1.75 liters during the very hardest work (such as requires a circulation of air up to 50 liters per min.). Hence, an apparatus that will permit the wearer to do any kind of hard work must supply not less than 2 liters of pure oxygen per minute.

Contrary to some assertions, it is useless to increase further the amount of oxygen in the hope of overcoming the harmful effects of carbonic acid in the air breathed. Unfortunately, oxygen does not have this quality, and its beneficial effects commence only when the air is free from carbonic acid. The development of the life-saving apparatus, since the first was constructed, about 20 years ago, shows a constant battle against the carbonic acid, which could not be done away with. Only an apparatus that will furnish its wearer with a circulation of air, always nearly free from this gas, will be thoroughly satisfactory. Careful experiments have shown the following:

Carbonic Acid Contained in	Parts per Thousand.
Bad room-air,	2
Unhealthy air,	10
Dangerous air,	over 20 to 25
Very poisonous air,	over 30
Deadly air,	40

According to these figures, a rescue-apparatus should never allow the amount of carbonic acid contained in the circulated air to exceed 10 parts per thousand.

Very interesting experiments were made by Mr. Draeger to ascertain: (1) the amount of carbonic acid contained in the respiration-air of different apparatus; (2) how this amount of carbonic acid affected the wearer of the respective apparatus; and (3) how far the working-power of the wearer was reduced on account of the presence of carbonic acid in the circulating air.

A number of tests, made under the supervision of a char-

tered chemist, have shown that a workman using an apparatus in which, during the two hours of working-time, the amount of carbonic acid did not rise over an average of 2 per 1,000, was able to do work equal to moving 35 tons through a distance of 1 m.; a workman using an apparatus in which the carbonic acid rose to 20 per 1,000 on an average, was

FIG. 1.—THE DRAEGER APPARATUS. FRONT VIEW.

obliged twice to be taken out of the smoke-chamber in a faint, and the work done hardly amounted to 20 tons. The work was registered by a machine, the workman lifting a weight of 25 kg. through a distance of 1 meter.

Experiments have further shown that the working-power of a man is reduced only 9 per cent. by wearing a Draeger apparatus, in which the quantity of CO_2 was only 2 per 1,000.

The foregoing shows that, besides a sufficient circulation of air and a sufficient supply of oxygen, the most important factor in the construction of a rescue-apparatus is the thorough absorption of the carbonic acid produced by the lungs. In fact, years of experience in working with rescue-apparatus in mine-fires and explosions has convinced me that an apparatus will be of

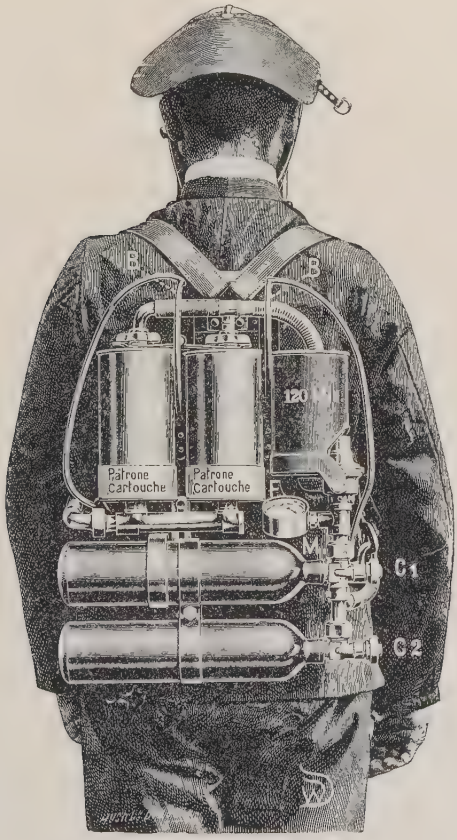

FIG. 2.—THE DRAEGER APPARATUS WITH TWO OXYGEN-CYLINDERS. REAR VIEW.

practical value only when its construction fulfills this requirement.

The weight of the different kinds of breathing-apparatus is about 30 lb. each, the variation being hardly more than one or two pounds, which, of course, is of little consequence. Of the lighter forms that are on the market none have met with success, on account of the insufficiency either of the supply of

oxygen or of the absorption of carbonic acid. So long as compressed gases cannot be stored in cylinders of smaller weight, or chemicals, such as sodium peroxide, used without producing sufficient heat to set the whole apparatus on fire (as was the case last year with a "pneumatogene apparatus" in Germany), a decrease in the weight of the apparatus cannot be obtained.

A feature in the construction of rescue-apparatus which should always be considered is the means provided for recharging. I consider the so-called "potash cartridge" of the Draeger apparatus, which comes packed and sealed, the best device for doing this work in the shortest possible time, and without any risk of mistake in the quantity of chemicals charged.

Of the helmet or mouth-bag devices adopted by different makers, the Draeger helmet outfit seems to me to be the most perfect, since it enables the wearer to breathe naturally and at the same time to hear, see, and talk with his fellow-workmen. Personal experience has shown to me the importance of this advantage.

A full equipment of the Draeger apparatus is now kept by the Boston & Montana Consolidated Copper & Silver Mining Co., Butte, Mont., for the purpose of fighting underground fire. After 20 years' experience with all kinds of apparatus, this company considers the Draeger to be the only one which has given perfect satisfaction. There are about 2,000 Draeger apparatus in use in various countries.

The complete apparatus, ready for use, is shown in Figs. 1 and 2. The main part, which is carried on the back, consists of a frame supporting the cartridge-holder, cooler, pressure-reduction valve, D, suction-nozzle, M, oxygen cylinders, C_1 and C_2, and pressure-gauge (finimeter), F. Steel guard-hoops, B, protect the apparatus from injury. Two metal tubes provide communication between the main part and the smoke-mask or respiration-bag. The apparatus proper is not soiled by use. The oxygen cylinders can be refilled easily, but the cartridges, after use, are replaced by new ones. Fig. 3 shows a similar apparatus with one cylinder, having parts cut away to illustrate the mechanical construction; the diagrammatic sketch, Fig. 4, outlines the operation. The potash cartridge (marked "Patrone Cartouche" in Fig. 2) contains 20 flat dishes so arranged

that the air to be freed from carbonic acid passes successively over each dish. The smoke-helmet, Fig. 5, consists of a pneumatic tube, L, shaped to form a tight joint with the head and face of the wearer. It extends from the chin to and around the top of the head. It is inflated by means of a rubber air-pump, G_1; a second valve, G_2, on the side of the helmet, being

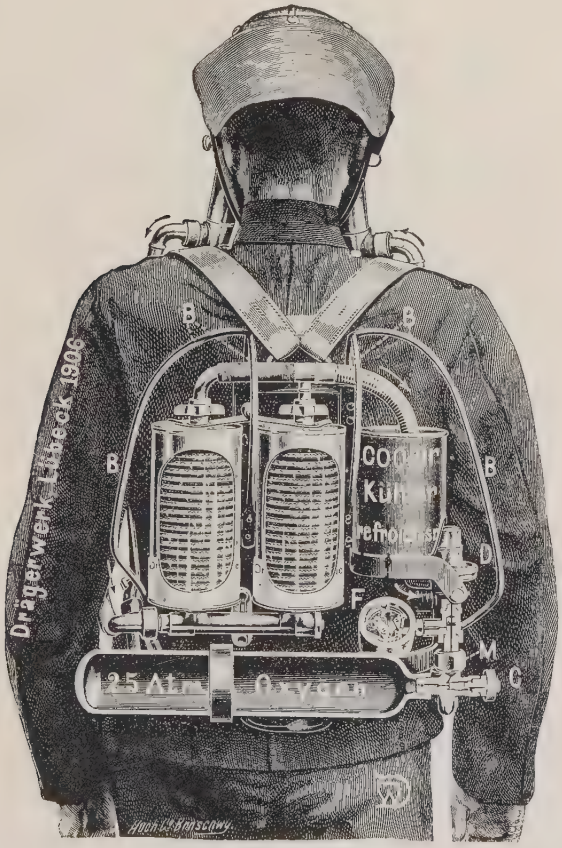

FIG. 3.—THE DRAEGER APPARATUS WITH ONE OXYGEN-CYLINDER.

provided to relieve the pressure if the pump is worked too rapidly. A leather strap with a buckle, Q, is provided for fastening firmly to the head. The respiration-bags are shown in Fig. 6. The bag at the right hand, S_1, serves as the pure-air reservoir. The air passes through the metal tube in the direction indicated by the arrow into the opening of the screw-joint, and is drawn in by breathing through the round tube of

the helmet. A small mica-plate valve in the tube (not shown in the illustration) allows the air to pass into the helmet, but prevents it from passing back into the supply-tube. The exhaled

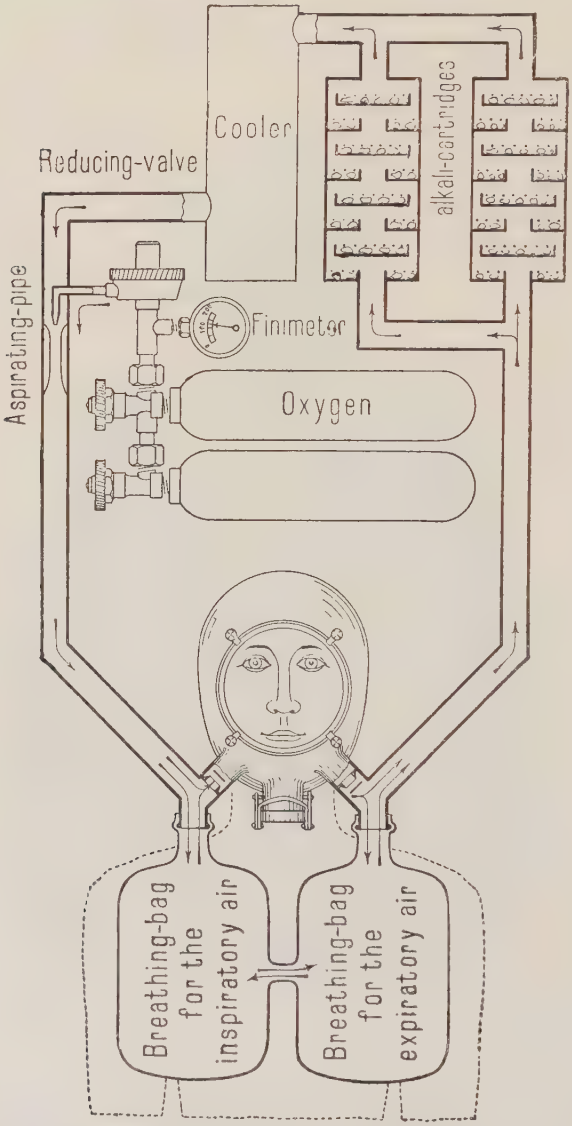

FIG. 4.—DIAGRAMMATIC SKETCH SHOWING THE METHOD OF OPERATION.

air passes through a similar tube into the second bag, S_2, having a mica-plate valve to prevent the air from returning to the helmet through an opening in the screw-joint, A. The two

bags are connected with a channel, S_3. The exhaled air is drawn through a wide metal tube to the cartridge. At the lower end of the helmet is a wide air-valve, *K*, for use while the apparatus is being put on. In this valve is a sponge with a projecting handle, *W*, which enables the wearer to remove any moisture collecting on the window of the helmet, or to reach the face. The bags are protected by a leather apron, shown in Fig. 1. The window is made of clear plates of mica

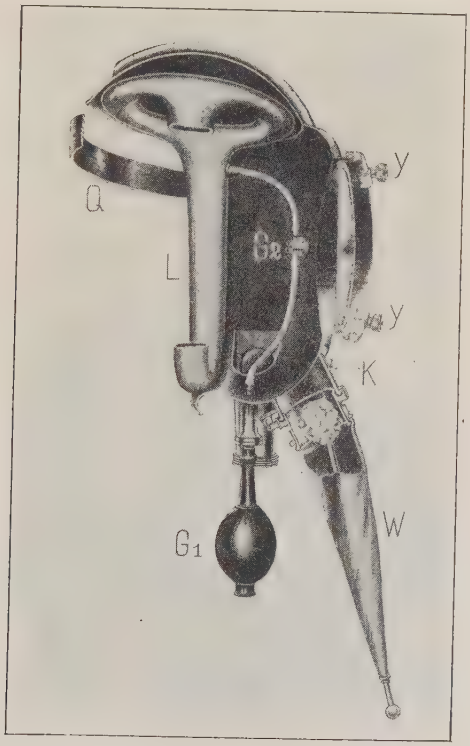

FIG. 5.—DETAILS OF THE HELMET.

or of glass, protected by a wire guard. It is held in place by four screw-hooks, *Y*, and can be easily removed.

Recent terrible mine-fires, explosions, etc., have called fresh attention to the subject of breathing-apparatus by the use of which spaces filled with poisonous gases, such as after-damp, can be traversed with safety for the purposes of exploration or rescue, or for direct fighting against fire.

It is perhaps not so generally known that it is not impos-

sible to resuscitate, by means of pure oxygen, persons who have been suffocated in smoke and after-damp, even when they have remained for more than two hours in these irrespirable gases. The Draeger works, at Lübeck, Germany, make an "oxygen reviving-trunk," which is kept on hand for this purpose at numerous mining rescue-stations throughout Germany. It consists of a cylinder containing oxygen, under very high pressure, connected with an automatic inhaler, and provided with a sufficient supply of pure oxygen for one half-hour's working. The apparatus is again made ready for use by simply putting

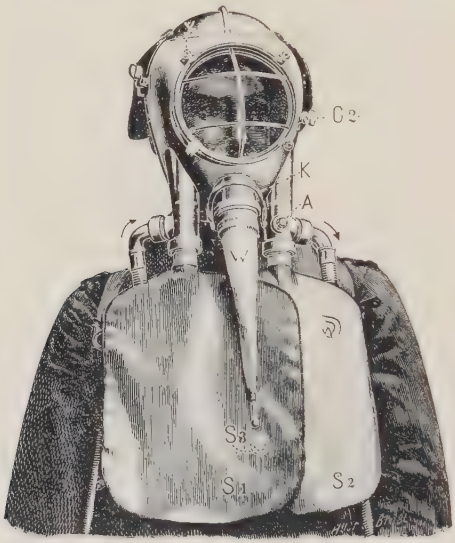

FIG. 6.—DETAILS OF RESPIRATION-BAGS.

in a newly-filled cylinder, which enables the work on a suffocated person to be continued for hours.

Persons, even after having been in after-damp for a considerable length of time, may be resuscitated by 10 minutes' treatment with pure oxygen. The official report of the German government mining officials, who investigated the Rheden disaster of Jan. 28, 1907, in which, out of 110 men underground at the time of the explosion, more than 40 were saved by the use of the Draeger rescue-apparatus, mentions that a treatment for about 10 min. was required to revive the suffocated.

The Ilsede Hütte Iron-Mines at Peine, Germany.

BY LUCIUS W. MAYER, NEW YORK, N. Y.

(Chattanooga Meeting, October, 1908.)

THE iron-mines of the Ilsede Hütte Co. are at a town called Peine, about 20 miles east of the city of Hanover, on the railroad to Brunswick (Braunschweig). Hanover, the capital of the province, is a military city, of approximately 250,000 inhabitants, and Brunswick, 33 miles to the westward, has half this number. The mines here described are located in a hamlet called Bülten.

The Ilsede Hütte Co. operates two mines, 1.5 miles apart, in this district, Gruber Bülten Adenstadt and Gruber Lengede Bodenstadt. It also operates an iron-mine north of Hanover, known as Eiserhaven, and one in the Harz mountains, the Georg Friedrich, at Grauhof.

At Bülten a coarse conglomerate, *a*, Figs. 1 and 2, contains nodules of limonite, cemented together by calcium phosphate. The ore contains Fe, 25 to 45; Mn, 3 to 4; P, 0.6 to 1.4; S, 0.02 to 0.09; and $CaCO_3$, 40 per cent. Directly above the conglomerate is limestone, *b*, followed by a mixture of clay and sand, *c*. Below the conglomerate is a hard clay floor, *d*.

The ore-body, from 10 to 12 km. long, and from 3 to 4 km. wide, has a maximum thickness of 20 m. It dips approximately 10° S.E., and strikes N. 45° E. Diamond-drill prospecting shows that the thickness diminishes to 1 m. as the end of the deposit is approached to the northward. At this point the deposit is 250 m. below the surface. In Fig. 2, the arrow, *A*, indicates the general direction of the work along the face, *a*, which is carried forward in a straight line for its entire length. At *B*, a train is approaching, which has come from the bench-track, shown at *C*, by a circuitous route, and is dumping the overburden behind the face, in the space left by the excavated rock.

Formerly the work at this mine was essentially open-cut, the deposit having a very strong outcrop. Figs. 1 and 2 show also

the depth that has been reached. During the past four years a large amount of underground mining has been done, which was necessitated by the thickness of the overburden, due to the dip of the ore below a practically flat surface. Figs. 1 and 2 show that the open-cut work has been carried forward to a point where the mine-plant is located, and it is now the intention to move these structures in order to make way for the mining operation.

At the time of my visit, October, 1907, the quantity of ore shipped to the furnace per month was 40,000 tons, of which 7,000 was mined underground. In course of time all the ore will be extracted by underground operations. The drilling is done by hand.

The regularity of the ore-body permits of methodical mining as shown in Fig. 3. A shaft has been sunk to intersect the ore-body at a depth of 70 m. When all the ore above this elevation has been mined, other shafts will be sunk beyond this point, and deeper ones. These will tap the ore at the lower elevations it attains with the gradual dip, and each shaft will control a new volume of ore behind it, up the rise.

The shaft-drift is run out in the direction of the strike of the ore-body, *B*, Fig. 3, and this drift prolonged about 300 m. At the 300-m. interval an incline, *D*, is started up the rise; and on each side of this incline, through which the ore is carried above to the haulage-road, *B*, are man-way inclines, *D'*, *D'*, cut somewhat smaller than the main inclines. At intervals of approximately 16 m. along the inclines, cross- or sub-level drifts, *E*, *E*, *E*, *E*, are run, and pillars, 8 by 16 m., are blocked out from this level. The chambers throughout are about 2 m. high and 3 m. wide. All of this work is on the foot-wall of the bed.

At *C*, Fig. 3, is a sump-drift, supplying sump, *A*, with clear water, from which level it is pumped to the surface.

At this place, where the bed is 8 m. thick, the method of work now followed involves two periods of three stages. The first period is the attack on the lower half of the bed. First, the pillars are blocked out. Second, the roof above the drifts, or chambers, is attacked to a certain height above the height of the original drift, corresponding to the height of pillars for this period. (For example, if a thickness of seam is to be

a, Coarse conglomerate iron-ore. *b*, Stratified limestone. *c*, Clay and sand.

FIG. 1.—OPEN-CUT METHOD OF STRIPPING THE FLAT ORE-BODY.

a, Coarse conglomerate iron-ore. *b*, Stratified limestone. *c*, Clay and sand.
d, Clay floor. *C*, Train transferring stripping.

FIG. 2.—OPEN-CUT METHOD OF BACK-FILLING THE STRIPPING.

FIG. 3.—UNDERGROUND METHOD OF MINING SAME DEPOSIT.

worked in two slices, and the drifts, as before mentioned, are 2 m. high, in a 4-m. slice, the roof will be brought down for 2 m. more before the final work of robbing the pillars is commenced.) Third, the pillars are robbed in conjunction with hydraulic sand-filling. When the ground in the lower slice has been entirely robbed and replaced by sand, this sand will be the floor for similar operations on the next slice above, which corresponds to the second period.

The method here followed, as compared with the work in the coal-mines at Carmaux, France, is to divest one slice completely and place it in proper condition for the working of the second slice before actually starting the work on this second slice. At Carmaux several slices of the same block of ground are worked simultaneously. As indicated in the lower part of Fig. 3, a section along one of the inclines, *D*, in the first period the ore is loaded into cars at the lower end of the pillars in each sub-level, *E*, *E*, whence it is trammed past the near man-way, *D'*, to the incline, *D*. Here the car is run on to a truck, which is hauled up and down the incline by rope-traction.

The inclined truck, which transfers the cars of ore from the sub-levels to the haulage-level, has a built-up level platform. When spotted at any sub-level, the cars are pushed on to this platform, the platform-track being so fixed as to meet the track of the sub-level. Thus, a car, as shown in Fig. 3, at *I*, passes on to the truck, *H*, and is lowered to the shaft haulage-way, *B*. At this junction the truck enters a depression in the haulage-road, and the car of ore is passed off the truck-platform and on to the haulage-road track, in the same manner as the car was passed on to the truck at the sub-level.

In the main haulage-road, electric locomotives gather the cars and conduct the train to the shaft. These locomotives, instead of having trolley-poles with a pulley-wheel arrangement at the end, are equipped with wide bar-iron rods in the form of a loop.

The dotted lines, Fig. 3, represent an area which is in the stage of being robbed. The work starts midway between the two inclines. A row of protection-pillars, *G*, *G*, 10 m. by 16 m., are left, which are finally removed before abandoning the block. This description applies also to the protection-pillars left between the main inclines and the man-ways.

The purpose of having a man-way on either side of the incline is to permit the men working in any block to pass down to the main level without going down an incline in which hoisting is being done, or crossing this incline in order to reach an outlet connecting with the main level. The work is started at the center, and there being an incline at either end of the block, the maximum tram at any stage of the operation in a block would not exceed 150 meters.

As before mentioned, the robbing-work will not be started until the roof has been brought down in the chambers around the pillars, to the maximum height of the slice. Following the withdrawal of pillars the area is filled.

The material used for filling is of similar appearance to that used in Upper Silesia, Germany, which is the seat of very extensive operations of this nature—a brown sand which stands very well after the water has drained off. The method of mining the pillars is in certain respects a counterpart of other work in the above district. The filling is brought in from above, being conducted through a bore-hole by the aid of water. Before allowing the sand to enter a chamber, posts are set at about 4-ft. intervals on one side of the chamber and against the wall of the adjacent pillar. Between this wall and the posts is placed 1-in. lagging, with scantlings set a few inches apart, but in no regular manner. This arrangement serves as a brattice to hold the sand intact, so that there will be no mixture with the ore when the work of removing the adjacent pillar is in its last stage. This method is very satisfactory, the pillars being robbed clean up to the sand, which is held vertically intact by the brattice.

The lower openings of the chambers are sealed in the ordinary way with posts and plank, forming a barrier to hold back the sand as it enters the chamber, and allowing the water to drain off. In the robbing-work, on reaching the upper end of the pillar, a 9-ft. stump is left, which serves as a safeguard, and is removed later on as the methodical work of robbing the pillars progresses.

The water, draining off rapidly from the sand, finds its way to the sump, *C*, Fig. 3, though in quite a muddy condition. Here the solution is dammed back until sufficient time has elapsed for the mud to settle, and the water is allowed to pass

over the dam and to the sump, *A*, from which level it passes on to the pumps.

Comparatively little timber is used in the present work. The roof overhead in the lower slice is ore, and when the lower slice is depleted the upper part of the ore-body will be in contact with the sand-filling of the lower slice. To what extent the conditions will be changed, as regards the amount of timber used, when the upper slice is taken, is interesting, for it appears, from the nature of the beds overlying the ore, *b*, Figs. 1 and 2, that an entirely different nature of roof-rock will have to be dealt with, and that much more timber will be necessary. The breaking of the ore itself, due to settlement, in the upper slice, will probably require less explosive than the quantity used in the lower slices.

The Silver-Mines of Mexico.

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(Chattanooga Meeting, October, 1908.)

Introduction.

The following general survey of the character and present condition of the silver-mines of Mexico, though not offered as a detailed treatise, has been prepared with care, is believed to be generally up-to-date, and is offered as a summary which may be found useful by mining engineers.

For several centuries Mexico has ranked first among the silver-producing countries. It is now second as a copper-producer, and seventh as a gold-producer, with a fast-increasing output every year. The total amount of capital invested in its mining- and smelting-works exceeds $250,000,000, while the value of the yearly metallic production is $160,000,000. There are more than 20,000 recorded mining properties.

The silver-mines of Mexico represent deposits of three principal classes: (1) veins and stock-works in eruptive rocks; (2) fissure-veins crossing the crystalline schists, granite, diorite, etc.; and (3) veins and replacement-deposits in limestone.

The true fissure-veins in some districts contain (in value)

more gold than silver. In other deposits, the silver is decreasing in depth, leaving galena and zinc-blende as the principal ores.

The following is a classification of the silver-deposits in the different States of Mexico:

In Eruptive Rocks.	In the Schists, etc.	In Limestone.
Western Chihuahua.	Eastern Sonora.	Eastern Chihuahua.
Western Sinaloa.	Zacatecas.	Eastern Sinaloa.
Western Durango.	Jalisco.	Eastern Durango.
Territory of Tepic.	Guanajuato.	Coahuila.
Queretaro.	Michoacan.	Nuevo Leon.
Hidalgo.	Mexico.	Potosi.
Puebla.	Guerrero.	Tamaulipas.
Lower California.	Oaxaca.	

I. Veins and Stock-Works in Eruptive Rocks.

1. *Western Chihuahua.*

The principal rocks are granite, syenite, and different kinds of porphyry. The surface, in some regions arid, is composed of recent volcanic rocks, like rhyolite, etc. The most important mining-districts are:

Batopilas and Andres del Rio, which contain irregular and low-grade ore-veins in diabase and porphyry.

Parral, including Santa Barbara and Minas Nuevas, a district in which mining has gone on for more than 300 years. The "Veta Colorada" is sometimes 90 m. wide between the quartz-porphyry and the schists, and has been traced along its outcrop more than 16 km. The famous Palmillas mine, although in this district, lies further west than the foregoing, in the limestone of the Sierra Almoloya. The ore-bodies are irregular bonanzas, reaching a depth of more than 200 m. The San Patricio and Rafael veins are parallel to the Veta Colorada. This district seems to be becoming richer in gold than in silver.

Mina is a district of quartz-veins included between the quartz-porphyry and the andesite. It contains the Guadalupe y Calvo and other mines.

Ocampo is becoming a gold-camp.

Arteago contains the well-known Palmarejo mine; Abasolo, the Cusihuiriachic mines; and Camargo, the Encinillas and others.

2. *Western Sinaloa.*

In this region, silver occurs in the districts of Fuerte, Concordia, etc., in veins bounded by porphyry and syenite. There are also quartz-veins carrying gold-bearing pyrites in the Sinaloa district (containing the Purissima, Pyramide, El Rosario, and Jesus-Maria mine), and in the Mocorito, Culiacan, and San Ignacio districts.

Sinaloa seems to be becoming characteristically a gold-producing region.

3. *Western Durango.*

Guanacevi is one of the most important districts. The mineralized rock is propylite (amphibolitic andesite). The deposit is a network of veins in all directions, with intervals of propylite more or less impregnated with iron sulphide. The total width of veins, including the rock impregnated, may reach 40 m., thus giving to the whole the form of a regular stock-work. There are no bounding selvages of clay. The ground is a zone of broken rocks, through which an easy access was open to the hot mineral waters. The mineral gangue is quartz and calcite, the calcite being apparently more favorable to the presence of silver. The feldspar of the porphyry seems to have been replaced, after decomposition, by the ore itself.

In some places the ore-deposit is an altered rhyolite, grayish or reddish, with intrusions of andesite dikes, containing fragments of rhyolite, and thus presenting a brecciated conglomerate. The green rock is reticulated with veinlets (called *hilos* in Mexico) of silver sulphide, so as to make stock-work. The minerals are pyrite, argentite, ruby silver, native silver, and gray copper, with some mispickel, galena, and zinc-blende in depth. The gangue is quartz and calcite.

The San Fernando district has some copper- and silver-deposits between walls of quartz-porphyry, andesite, and rhyolite.

The districts of Gabriel, Conatlan, and Rodeo contain some rich old mines of silver and some copper in depth. (The rich mine of Avino, which gave birth to the cathedral of Durango, is being reopened at the present time.)

Topia and Tepehuanes show some lead and zinc in depth, below the old workings.

The district of El-Oro-Inde produces more gold than silver.

4. *The Territory of Tepic.*

This region contains some very irregular silver-veins.

5. *Queretaro.*

The silver-veins of this region are in process of slow development.

6. *Hidalgo.*

One of the most famous mining-districts of Mexico is in this State. Pachuca and Real del Monte are known to have been localities of mining, even before the landing of the Spaniards in Mexico.

The ore-deposit is an andesite, intermixed with quartz-veins, rich from the surface. The country-rock around the andesite is dark-green quartz-diorite, rhyolite, and basalt, with slates and argillophyres. The long and wide main vein at Pachuca is a true fissure-vein, nearly similar to the Comstock, in Nevada, or the Schemnitz main vein, in Hungary. The green rock, formerly called trap, is similar to the trachytes or propylites of Schemnitz. This great vein was rich from the surface, and gave some bonanzas below 150 m. in depth.

On the Santa Gertrudis vein, 4 m. wide, with walls of pyroxene-andesite, the workings have reached the depth of 400 m. The San Rafael, Sorpresa, and Soledad mines are worked on a net-work of smaller veins, yielding from 400 to 500 g. of silver per ton, at a depth of 400 meters.

The Pachuca and Real del Monte ores are native silver and stephanite, with a gangue of quartz and some calcite and very little barite. The sulphides of copper and zinc are rare.

Pachuca is the last refuge of the *patio* process, in which the mules were finally replaced by electric plows. But now there may be seen, at the San Francisco mill, the most modern plant for cyaniding, including conical tanks 42 ft. high, with inner cylinders for agitation.

7. *Puebla.*

The silver-veins here are, at present, of little importance.

8. *Lower California.*

The veins here are mostly of auriferous quartz. But some silver-veins in granite and syenite, accompanied with dikes of porphyry, have been found at Triunfo.

II. Fissure-Veins in Granite, Schists, Etc.

1. *Sonora.*

The silver-mines, situated in the districts of Magdalena, Alamos, and Moctezuma, have shown in former times some famous bonanzas; but operations are now decreasing. The characteristic of these deposits is a sudden enrichment at the crossing of veins, which beyond such a crossing almost completely disappear.

2. *Zacatecas.*

The mines of Zacatecas and Fresnillo are well known. There are some very extensive fissure-veins, often from 20 to 30 m. wide, crossing the schists, the quartzites, and even the limestone, then the *grauwacke*, then a red conglomerate with fragments of syenite, and lastly a sedimentary tufa with pebbles of porphyry (not volcanic). The zone of bonanzas is the sulphide zone, below that of the oxides and chlorides, and above that of the antimonial ores, and therefore ranging between 100 and 300 m. in depth.

The miners commonly give the name *colorados* to the red oxides, the chlorides, and bromides; *negros* to the sulphides, antimonides, and arsenides; *azulaque* to the iodides and tellurides (*plata azul*). (The term *azulaque* is applied elsewhere in Mexico to material containing finely disseminated ore, or stained with the products of ore-decomposition.—R. W. R.)

The total silver product of Zacatecas from 1548 to 1867 has been estimated at $800,000,000 (U. S. gold), excluding stolen ores and ores not taxed. The average value of the ore has been estimated at $35 per ton, probably after some rough sorting.

The quartz-veins at the crossings of El Bote reach 16 m. in width, at the present depth of 280 m. The bonanzas were from 40 to 60 m. long.

A considerable number of such veins have been opened at Zacatecas, extending as far as from 10 to 15 km. in length, but crooked and irregular. Sombrerete is a continuation of the same veins. They have decreased much in importance at present; but some new openings of copper-ores in lenses have been made, carrying gray copper with galena and zinc-blende between walls of limestone and granite or porphyry.

3. *Jalisco.*

This district exhibits bodies of complex ores, carrying copper, silver, gold, lead, and zinc in the schists and in a conglomerate with pebbles of dolerite. Dikes of porphyry intrude with much irregularity.

In the Mascota district, chlorides and tellurides occur.

4. *Guanajuato.*

This well-known mining-camp has been much developed recently with the aid of American capital. The mines had been carefully worked as deep as 400 and 500 m., but the low-grade ores had been left intact, together with the residues of old treatment, or *terreros*, and the resources of deeper levels.

The more economical cyanide process seemed to solve the problem of the low-grade ores, even with arsenical and antimonial ores; and seven large mills have been erected. The total present production (March, 1908) is between 10 and 12 tons of silver monthly. Development-work is slowly advancing.

Guanajuato is credited with the same production as Zacatecas, $800,000,000. The three main veins are 6, 7, and 12 km. long. The largest is in some places 100 m. wide. It crosses the hornblende-schists, the *grauwacke*, some dikes of diorite, andesite, serpentine, and a stratified conglomerate of feldspar tufa, locally known as *lezero*. The quartz is often violet in color. The ore is usually oxides and chlorides, then stephanite, argentite, and polybasite, with some iron and lead sulphides. There has been little gold. The gangue comprises quartz, gypsum, fluorspar, siderite, asbestos, apophyllite, and calcite. The average value of the ore has been $17 (U. S.) per ton.

The La Luz district contains the famous mines of Providencia and La Luz. The bonanzas are mainly in the talcose schists. At a greater depth, in the black aluminous schists, the veins are barren.

The three most famous mines of Guanajuato have been the Valenciana, the Mellado, and the Rayas. The most productive mill at present is the Pinguico.

5. *Michoacan.*

The most famous district is Angangueo, from which, for more than 50 years, a group of 13 mines, all belonging to the

same owner, shipped more than 200 tons daily to the smelters of Aguas Calientes.

A revival of prosperity has occurred at El Oro, in the vicinity of Angangueo, through the operation of three large mines, El Oro, Esperanza, and Dos Estrellas. A fourth, the Mexico, has begun shipments. These are commercially rather gold- than silver-mines. The veins are very wide, and carry quartz (mostly "sugary") with gold and argentite, between walls of schists. There has been a regular bonanza in the first vein of Dos Estrellas. The other veins vary in width from 6 to 20 m., of quartz, generally soft. At El Oro there are some stopes 40 m. wide. As the walls are liable to cave, square-set timbering is generally used in the stopes. The average proportion of gold to silver in the ore is 16 g. gold to 100 g. silver; but since the silver-extraction is less complete than the gold-extraction, the proportion in the ingots is 25 g. gold to 100 g. silver.

The district of Tlalpujahua was made famous by the mines of Jean de la Borde, who exploited four rich bonanzas, working, without steam-engines, to the depth of 350 meters.

6. *Mexico.*

The principal mining-districts are Temascaltepec, Sultepec, and Zacualpan. The mines are quartz-veins in the crystalline schists and the granite, with frequent intrusions of igneous dikes.

At Temascaltepec there is a net-work of veins running E–W. and NW–SE., containing chimneys, lenses, and ore-shoots (*clavos*) of irregular shapes. The intervals between these bonanzas are either low-grade quartz, several meters wide, with barren "horses," or mere veinlets (*hilos*) of quartz, scarcely visible. It has been possible, however, to trace some veins from 6 to 7 km. The pockets and chimneys extend horizontally from 1 or 2 to 30 or 40 m., but the shortest ones are the richest. In thickness they reach 4 to 5 m. Sometimes the total value of such a bonanza is only from $5,000 to $12,000; but there have been some which yielded from $400,000 to $600,000 (U. S.). Since 1885 the Rincon and Quebradillas mines reached the last figure. These districts are characterized by the bonanzas rather than by an abundance of low-grade ores—a circumstance which explains the enormous amount of old workings down to 250 m. in depth.

The most important mines have been Mina de Agua, El Rincon, El Socorro, Quebradillas, etc. Almost all of them have had to deal with large masses of water coming from cross-veins or dikes.

The El Rey, or Guitarra vein, is a large lode of quartz, 15 to 20 m. wide, in granite. In value it is irregular, running from 100 to 400 g. of silver per ton, but with pockets of rich ore. Quicksilver was predominant on the surface—so that a quicksilver-furnace was erected for its production. The general course of veins is irregular. *Quebradillas* means fractures. The El Rey vein seems to run towards El Oro.

The cyanide process is in course of introduction at Temascaltepec for the treatment of antimonial ores, which are refractory to amalgamation. The result for the first tonnage crushed reached a total extraction of 70 per cent.

Sultepec is a district of quartzose schists and metamorphic slates, uplifted by trachytes. There are no clay-gouges bounding the deposits, which show a progressive impregnation for 1 to 1.5 m. In some places there are 6 m. of quartz and kaolinic clay. Mostly predominant are argentite and pyrites: there is very little galena or zinc-blende. The deepest shaft reaches 150 m. The most recent bonanza (1903) yielded $300,000 (U. S.).

In the Zacualpan district occur similar quartz-veins in the schists, still more numerous, and liable to exhibit bonanzas.

The three districts of Temascaltepec, Sultepec, and Zacualpan are credited with a production of $60,000,000 (U. S.). The veins are smaller than at Zacatecas, Guanajuato, and El Oro, and have been worked to a smaller depth. Probably the less extensive stopes and the more abundant water have been serious drawbacks. The mineralized zone at El Oro differs somewhat from the others in the nature of the quartz, the presence of gold, and the more regular occurrence of low-grade ores.

7. *Guerrero.*

Here the formations are much entangled. The ores are either in the slates and the volcanic rocks, or in the Cretaceous limestone; dikes of andesite protrude everywhere.

The mineral zone of Taxco presents a complex set of silver-, antimony-, zinc-, and tin-ores. Sometimes the veins are from

1 to 2 m. wide. The old bonanzas of Taxco, worked by Laborde, yielded more than $60,000,000 (U. S.).

The most important bonanzas are in the limestone; but the largest masses of ores are in the syenite and andesite, near the boundary of the State of Mexico, in the Vadelista mountains.

There is some copper and gold in Guerrero; but the greater part of this State is uninhabited and unknown, though ancient ruins speak of a former population. The climate is good and water is plentiful.

8. *Oaxaca.*

In Oaxaca the geological formations are still more complicated than in Guerrero. The Zempoaltepec mountains are like a knot of the big chains of Mexico. There are more than a dozen mineral zones, all irregular, and containing silver, gold, copper, lead, zinc, etc. The old silver-workings are very extensive.

III. Ore-Deposits in Limestone.

1. *Eastern Chihuahua.*

The Cretaceous limestone of this region contains some of the largest silver-ore deposits in Mexico. This limestone, with its caves and water-courses, is peculiarly favorable to ore-deposits, often of secondary origin, and therefore very rich, as the products of a second concentration.

The district of Jiminez contains lead carbonate ores, with brittle silver, ruby silver, and native silver, replaced by copper in depth. This includes the limestone of the Sierra Almaloya, with the rich mines of Palmillas, Cigarrera, Tacuba, etc. The Palmillas is the well-known mine of Pedro Alvaredo, which has been leased by an American syndicate.

The great silver-deposit of Chihuahua is the Santa Eulalia, near the town and railroad of Chihuahua. The old workings are said to be of Aztec origin, before the coming of the Spaniards, and the total production since 1705 has been estimated at $200,000,000 (U. S.).

This deposit is in a mountain of limestone with deep ravines, covering a surface of from 10 to 12 sq. km. The ore is in chimneys, pockets, and caves of all kinds. The present lowest stopes, 500 m. below the surface, show the same ores as occur

above: chloride, bromide, iodide, and sulphide of silver, lead carbonate, galena, and iron oxides.

The limestone is stratified, and forms an anticlinal fold with a gentle slope (from 5° to 16°) N–S. In places it passes into dolomite. On the summit there is a volcanic tufa, below which the limestone is often mineralized. The porphyry dikes are of only occasional occurrence and are narrow, but become more important in depth.

The mineralized zone is from 750 to 1,000 m. wide. The ore-deposits are of two kinds: the *mantos*, or flat replacements of limestone; and the *abras*, or fissures, more or less vertical. The ore is either contemporaneous with the cavities, or posterior to their opening.

In the upper cavities, the lead- and silver-ores have been dissolved, and again precipitated in the lower cavities (from 300 to 400 m. below the surface), thus forming new bonanzas richer in lead carbonate and in silver. Some of these caves attained a width of from 80 to 100 m. The Bull-Pen cave, 390 m. from the surface, has a diameter of 130 m. and a height of 30 m. Narrow channels and fissures join the main caves. In one mine there is a natural water-course descending (as measured vertically) 210 m. Millions of tons of limestone have been leached by the water during the past centuries. There is, in one instance, a curious helicoidal channel between two big caves, which facilitates the ore-transportation. The mines encounter very little water. The deepest mine, the Santo Domingo (520 m.) is still in carbonates. The richest mines thus far known are in the middle of the limestone-deposit. There is very little gold, and that in a few mines only.

2. *Eastern Sinaloa.*

The silver-mines in limestone, thus far known, exist mainly in the Rosario district. The limestone is intersected with porphyry dikes. Sinaloa is more conspicuous through its gold-bearing quartz.

3. *Eastern Durango.*

There is some silver in limestone at Canelas, Lerdo, and Trinidad; but the richest district is Mapimi, near Torreon, which contains the Peñoles mine. This is a deposit in limestone intersected with porphyry dikes. The ore consists of

oxides and carbonates, with very little copper and zinc. As at Santa Eulalia, it occurs in irregular bonanzas. There is a tunnel which crosses a dozen veins. The present maximum depth of working is 650 meters.

4. *Coahuila.*

The Sierra Mojada contains, in the Carboniferous limestone, some silver-bearing galena in the form of pockets and impregnations. There is also some lead carbonate and zinc carbonate in replacement-deposits. Saltillo is the richest district.

5. *Nuevo Leon.*

In this State the production of silver is, at present, decreasing.

6. *Potosi.*

In this State the production of silver is decreasing, at present, in the districts of Charcas, Ramos, and San Francisco de Guadalcazar.

The richest district is Catorce, which is credited with a production of $500,000,000 (U. S.) from workings not exceeding 250 m. in depth. The limestone here overlies andesite, and is covered by rhyolite. The ores are contact-deposits, between andesite and limestone; the rhyolite is barren. The minerals are silver chloride, lead carbonate, and hematite, with a gangue of calcite and chalcedony. There is some gold in the hematite. Copper, galena, zinc-blende, and iron sulphides occur. The ore-bodies are of great size, but irregular. There is no map of the old workings in this camp, and new development-work is consequently difficult. Where gold occurs, there is no native silver. The reverse is true also. The ores cannot be, at the same time, of high grade in gold and in silver.

At the present depth at the Santa Anna mine (600 m.), the iron sulphide is increasing. The old workings have a total horizontal extent of 40 km. The ore yields sometimes 3 to 4 kg. per ton, mostly from silver chloride and *plata azul.*

IV. Ore-Treatment.

The old *patio* process was still recently in use at Pachuca, with some modern improvements. But even there it is giving place to the cyanide process, which, having been applied to

gold-ores for the past 12 or 15 years, has needed little modification to make it available for the treatment of silver-ores. Yet there was a profound difference to be considered. In the gold-ores treated by this method the gold was usually free; whereas, in refractory silver-ores, the silver is combined with sulphur, arsenic, and antimony. It was therefore necessary to reduce the silver first. This result was obtained partly by fine crushing in tube-mills, partly by agitation with compressed air, centrifugal pumps, etc., or by simple mechanical agitation.

In the old *patio* process the reduction was obtained by the copper-sulphate (*magistral*) and sodium chloride forming iron and copper chlorides, and afterwards replacing sulphur, antimony, and arsenic in the silver combinations by silver chloride, easily amalgamated with quicksilver.

In like manner, the leaching-power of potassium or sodium cyanide is increased by the addition of another metal (iron, zinc, or lead acetate), which would replace the silver in its combinations, and thus permit its solution in the cyanide. Compressed air seems to be a powerful agent, as was the *magistral* in the *patio* process; but air alone gives almost the same result. Agitation is mainly necessary for breaking up the clay-balls of the slimes, in order to put the ore in contact with the cyanide.

The *patio* process gave an output of 95 per cent. of the silver and 84 per cent. of the gold contained in the ore. The cyanide treatment saves from 90 to 98 per cent. of the gold. As regards silver, the results, different with different kinds of ores, run from 50 to 80 per cent.—in some cases even 85 per cent. The total process of concentration and cyaniding yields from 70 to 90 per cent. Preliminary experiments give a fairly good idea of the probable results in each case. But it should be added that further improvements in the cyanide treatment are still to be expected.

The Mining and Milling of Silver-Lead- and Zinc-Ores at Pierrefitte Mines, France.

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(Chattanooga Meeting, October, 1908.)

I. History.

The Pierrefitte mines, situated in the South of France, in the district of the Hautes-Pyrenees, owe their name to the fact that the first mining operations of any extent, and of comparatively recent date, were carried out in the neighborhood of the village of Pierrefitte, which is a branch terminus of the Midi railway.

The Pierrefitte mines, however, embrace a wide extent of country and several different concessions, on most of which comparatively little exploration-work has been done. The portion of the property presently to be described is a zone running roughly E–W., over a distance of about 5 miles from the valley of Cauterets to the valley of Estaing, the ridge dividing these valleys being a spur of the main chain of the Pyrenees, and running more or less at right angles to it.

According to local tradition, which various indications tend to confirm, these mines are of ancient date, the first workings being attributed to the Romans, who are supposed to have sought not only the lead but also the silver which is contained in the ore. Old slags have been found in the neighborhood which might well have been produced from the cupellation of lead-ores.

These old workings, while of considerable extent at times, do not seem to have extended beyond a depth of 100 ft., even this depth being somewhat exceptional.

The old notarial records of the district show that mines were known to exist, and that they were even worked, in the sixteenth century, but since the mines were abandoned (by the Romans or others) considerably prior to this period, very little

* Consulting Engineer.

was done until the Marquis de Querrieu took up the concessions about 60 years ago.

Operations were started on a small scale, but it was not until an English company purchased the titles from the Marquis that mining on a more or less serious basis was commenced.

Practically, the only part of the property exploited during the period of more than 40 years was the Pierrefitte mine. During this period, in spite of the discontinuity of the records, it seems certain that approximately 500,000 tons of ore were mined.

The English company, above mentioned, erected a milling-plant of a capacity of about 40 tons per day, and put up a cable connecting the mill with the levels of the Pierrefitte mine. This company, for a time, was fairly successful in its operations, having made during three years sufficient profit to pay dividends exceeding £47,000, and also to develop, to some extent, another lode known as "Estaing," about 3 miles west of the Pierrefitte mine, show in Fig. 1. As, however, practically no development had been carried on ahead of the stopes in the Pierrefitte mine, its only source of ore-supply—the high-grade ore-supply—which it had secured at the commencement could not be maintained; and this circumstance, combined with the fact that the mill was getting into a bad condition owing chiefly to insufficient outlay on renewals and repairs, brought the company into an embarrassing position, from which, in consequence of its having paid away all its profits in dividends, it could not extricate itself, and in the year 1892 the mill was shut down. During the following five years practically no work was done except a small amount of development in the Pierrefitte mine, by which it was hoped to meet again with the large bodies of high-grade ore which had been lost.

Being unsuccessful in this work, the whole property was offered to and taken over by a small syndicate, which worked the mines under a lease and option of purchase for two years—1898–1899. This syndicate purchased the property, and was converted into a company Jan. 1, 1900, called the Mines de Pierrefitte, Limited, the present operating company.

The shareholders of the syndicate became the shareholders of the present company, and the continuity of operations may, therefore, be said to have been unbroken from the commencement of 1898 down to the present time.

The present company remodeled and overhauled the old mill, which had fallen into complete disrepair, erected a plant on the Cauterets river for the generation of electric power, and carried out considerable development-work in the Pierrefitte mine. For several reasons, however, this last operation did not meet with success, and, with a view to procuring new supplies, the Estaing lode was for a short time actively developed and about 35,000 tons of good pay-ore opened up there. Owing to want of funds to meet the cost of a hauling-plant to deliver the ore at Pierrefitte, this supply was not, however, available for milling; and as a period of great depression in the metal market had in the meantime set in, the company, which had from the start been meeting the difficulties of the position with insufficient working-capital, found it necessary to abandon practically all development, and for two years prior to my principals becoming interested in the property the mill had been run entirely from supplies of ore drawn from reserves opened up by previous development.

This state of things naturally could not be long continued, and in the year 1905, when the ore-reserves had practically come to an end, the shareholders agreed to part with the larger portion of their holdings in order to provide the capital which my principals have raised for the new plant and development, the total cost of which has exceeded £50,000.

II. The Estaing Mine.

On making my first examination and report on this property, about three years ago, my attention was directed chiefly to the Estaing mine, owing to the ore-values and the lode being better adapted for rapid development, together with the simple nature of the ore, which would allow of higher recoveries than that of the old Pierrefitte mine. .

The Estaing mine is situated on the western side of the divide, about 2,000 ft. above the village of Estaing, and distant from it about 2 miles by the mule-trail.

1. *Transportation.*

It was formerly proposed to put up a rope-way to carry the ore down to a mill to be erected in the valley, whence the concentrates could be carted to the station of Argeles, 9 miles

away, but for many reasons I considered this inadvisable, and advocated the erection of a rope-way to carry the ore to the existing mill-site at Pierrefitte. Briefly, the principal reasons for my decision were the following:

1. The road from the village of Estaing to the Argeles station is narrow, difficult to keep in repair, and subject in the winter to frequent stoppage from snow and landslides; moreover, in order to widen it sufficiently for continual traffic, a large amount of money would have been required to compensate the wayside land-owners.

2. The erection of a mill at Estaing would be attended by considerable cost for transport of material, and by a large outlay for quarters for the men, since there is absolutely no accommodation in the village. Also, owing to the comparatively high altitude (3,000 ft.) as compared with Pierrefitte (1,800 ft.), the conditions for uninterrupted work are much less favorable.

3. Finally, in favor of the rope-way across the mountain to Pierrefitte, there was the fact that the most favorable route passes in proximity to nearly all the intermediate outcrops, some of which are of considerable importance.

The capacity of the rope-way permits the holding of a large reserve to be used in case of temporary stoppage during gales; and the ore can always be milled and the concentrates conveyed to the Pierrefitte station by an electric railway in all weathers. The rope-way will be described later.

2. *Lode-Conditions.*

The formation in which the Estaing lode is situated consists chiefly of Silurian schists, the lode occurring at the contact between a fairly even stratified schist and a black shale generally more or less foliated.

The lode follows approximately the stratification of the gray schist, but there are exceptions to this, and it also occasionally happens that the lode is entirely in the gray schist.

There are frequently several parallel zones of the gray and the black schists, generally with a lode in the contact, the width of the lode varying from a few inches to as much as 50 feet.

The metalliferous values are comparatively evenly distributed throughout the lode, though at times the richer mineralized portion of the lode follows a somewhat tortuous course. The

lode-filling is composed chiefly of schistose matter and quartz. A small amount of pyrites, sometimes cupriferous, accompanies the ore. The average assay-value is: zinc, 13; lead, 6 per cent.

There are, roughly speaking, two lodes or two spurs of the same lode running E–W., and dipping north, at an angle of about 70°, on which the development-work has been carried out, but there are various other seams and stringers of less importance, which may pay for stoping. These lodes are indicated on the surface by strong outcrops, which extend visibly some distance down the slope of the mountain, of which the ridge is formed by the principal outcrop. The lode is attacked from the surface by three main levels, distant 150 and 185 ft. apart respectively, and referred to hereafter as Nos. 1, 2, and 3, No. 3 being the adit. There are two intermediate levels between 1 and 2, and 2 and 3, respectively, and a rise from the No. 1 level to the surface, whence another level has been started 95 ft. above No. 1. The vertical distance from the end of the No. 3 level to the outcrop is 620 ft.

3. *Mining-Methods.*

Owing to the somewhat tortuous course followed at times by the mineralized portion of the lode, it is not practicable to adopt a uniform method of stoping, both the overhand and underhand methods being employed, as the occasion and the existing exploration and development-work may render advisable, but in the great majority of cases the overhand filled-stope system is adhered to.

The walls are firm, as a rule, and but little timbering is required in either the levels or the stopes. The ore is passed through chutes to the double-track adit-level, where it is loaded into cars and trammed to the loading-station of the rope-way.

At the present time, hand-drilling is employed exclusively, though it is possible that it may be supplanted by machine-drilling later on. The miners are mostly Spaniards or Basques. They drill double-handed, and prefer an exceptionally heavy hammer, most of them asking for a 10-lb. and none for less than a 9-lb. hammer. As a rule, they do not care to attempt any change, and do not take kindly to single-handed drilling.

Two men on contract, working double-handed, drill from 6 to 7 ft. of hole per shift of 10 hr. in fairly hard rock.

The mine is much exposed to the wind, and is surrounded by snow for several months in the year—eight months in 1906–7. The temperature often falls well below 0° F. in January and February, and, owing to the prevalence of wind, the surface-men occasionally suffer from frost-bite. In consequence, the question of properly housing the men became urgent, and a barrack was constructed which was the outcome of considerable study of the conditions of the people. Transport being very expensive, every effort was made to economize in the amount of material not to be found on the spot. The walls were made in dry masonry with mud filling, only the corners and the openings of the building being built with lime-mortar. It has two stories, the lower floor consisting of the mess-room, kitchen, and store-room, and the upper floor of the dormitories. There are four lines of bunks in double tiers running the full length of the upper floor, with the exception of a small space partitioned off for the shift-bosses. There is a lavatory, etc., on the upper floor, and a lavatory and lamp-room on the lower.

The windows are iron-framed, and have a hinged panel which can be screwed down on a rubber gasket like the port-hole of a ship, and can also be fixed at any desired width of opening.

The building is lighted throughout with acetylene gas drawn from a central generator. The miners also use acetylene lamps in the mine, and these give very satisfactory results, as regards both general convenience and cost. It is found that for a much stronger light they cost little more than half as much as the colza-oil lamps usually employed in France.

III. The Rope-way.

The rope-way, constructed on the "Otto" system, has a capacity of 200 tons per 10 hr. It is 4.2 miles in length, with three angle-stations, and a difference of level between the loading- and unloading-stations of 3,000 ft. Fig. 2 is a view of the line crossing the Col de Pierrefitte, and Fig. 3 shows the end of the line entering the new mill.

Three independent engineers examined the proposition with a view to tender for the contract on behalf of their respective firms, and all selected the same route as the only one practicable, the topography of the country rendering the three angle-stations absolutely necessary. In consequence, the cost of the

FIG. 1.—OUTCROP OF THE ESTAING LODE.

FIG. 2.—ROPEWAY-LINE CROSSING THE COL DE PIERREFITTE.

FIG. 3.—THE NEW PIERREFITTE MILL (LOOKING SOUTH).

rope-way per mile was comparatively high, the short summer and the difficulties of transport being also important factors in the total expenditure.

There are 63 standards, varying in height from 3.50 to 31.50 m., also an intermediate anchorage and an intermediate tension-station. Fig. 4 shows the profile of the line, but does not, of course, indicate the lateral slope of the ground, which is in places considerable. Fig. 5 shows the plan and elevation of the charging-station, and Fig. 6 of the discharging-station. The maximum gradient of the line is 28°. The longest unsupported span is 630 meters.

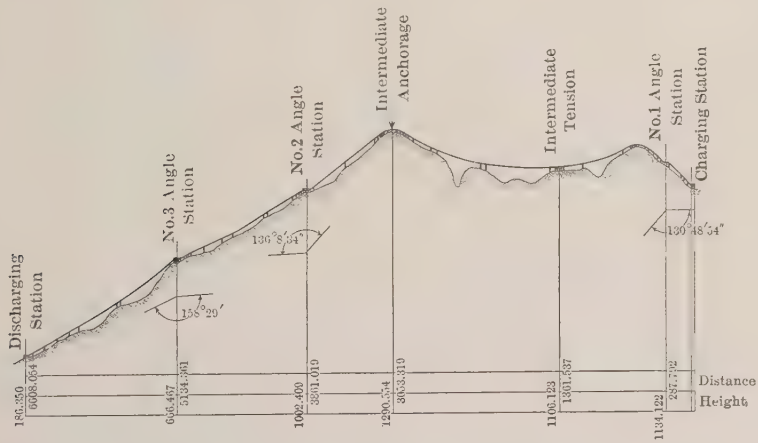

Fig. 4.—Profile of Aërial Ropeway of Mines de Pierrefitte.

The capacity of the buckets is about 325 kg., the speed of the hauling-rope is 2 m. per sec., and the distance between the buckets is approximately 108 meters.

The line is automatic with 100 buckets, and with the full number of 135 buckets will have an excess of hauling-power of 28 h.p., which is absorbed by a hydraulic governor in the shape of a closed, reversed turbine at the third angle-station; but for the purpose of starting the line and of re-establishing the excess of load on the down side, if for any reason the normal state of things were disturbed, a 25 h.p. "Otto" gasoline-engine was installed at the first angle-station.

The angle-stations are not automatic, as it was considered that in view of the difficulties presented by the line consequent upon the profile, the number of angle-stations, and the climate

and general conditions, any additional complication might prove a doubtful advantage.

The loading-station forms a terminal loop to the line, and contains no mechanism beyond a sheave for the hauling-rope. It is provided with six bins, three on each side of the loop, into which the cars from the mine are tipped. These bins, of a total capacity of 200 tons, are fitted with chutes on the

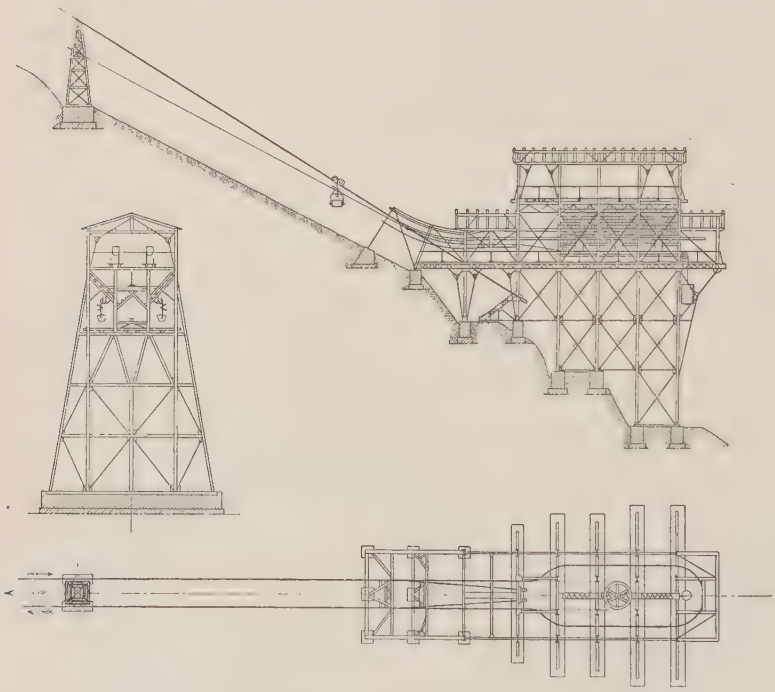

Fig. 5.—Charging-Station, Elevation and Plan.

hinged-spout principle which load the ore into the rope-way buckets.

The No. 1 angle-station contains, as before mentioned, a gasoline-engine suitably geared to the hauling-rope sheaves, and which can be immediately thrown out of action by two independent clutching-devices. It also contains two independent brakes, each sufficiently powerful to stop the whole line.

The intermediate tension-station, the intermediate anchorage, and the No. 2 angle-station are of the usual type, and contain no mechanical parts beyond the guide and counter-weight

sheaves, the hauling-rope being continuous from the No. 1 to the No. 3 angle-station.

At the No. 3 angle-station the mechanical arrangements re-

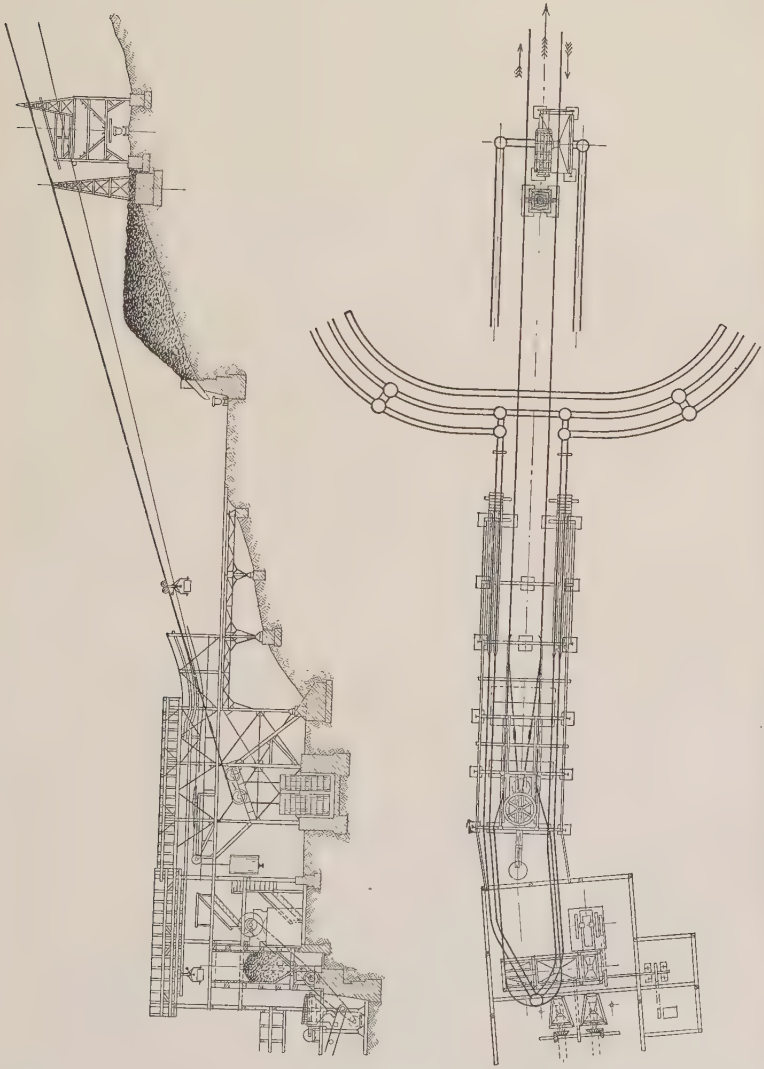

FIG. 6.—DISCHARGING-STATION (ELEVATION), SHOWING SUPPLEMENTARY ARRANGEMENT (PLAN) FOR DISCHARGING ORE TO THE STOCK-PILE.

semble those of the No. 1 station, except that in place of the motor there is the hydraulic absorber-brake, which automatically regulates the speed of the hauling-rope to within 3 per cent.

The unloading-station forms the other terminal loop of the

line, the buckets passing over the top floor of the mill, as will be described later. There are shunt-rails to keep spare buckets or to relieve the line and to load for the up-traffic when necessary.

At the last standard before reaching the unloading-station there is an automatic tipping-device which dumps the buckets, when required, into a bin with a chute, whence the ore can be trammed to the reserve-dump disposed radially around this point. From the reserve-dump, which is kept as nearly as possible at 4,000 tons, the ore can be trammed from various chutes arranged along the face of the dump direct on to the platform of the unloading-station by means of a bridge.

The diameter of the bearing-ropes is 28 mm. on the loaded side, and 20 mm. on the return side. The ropes are of the spiral construction, without strands, and their breaking-strain is 140 kg. per sq. mm. The hauling-rope has a uniform diameter of 22 mm. It is of the ordinary type with hemp core, and is of steel, breaking under 180 kg. per sq. mm. The grips, of cast-steel, on the carriers, are locked and unlocked from the hauling-rope automatically, and will work safely on inclines much greater than the maximum on the present line, which is about 28°.

All the stations are connected by telephone, the telephone-wire being mounted on posts quite independent of the rope-way. On the most exposed portion of the line, the wire is taken underground by a lead-covered cable.

The standards and stations are all built of steel, and are erected on masonry foundations, to which they are anchored by rails and eye-bolts. The station-roofs are of galvanized iron, and in addition to being screwed down, are fastened by wire-ropes running over the top, as the wind is often very high.

The flooring is of plank, with the exception of that above the machine-rooms in No. 1 and No. 3 stations, which is of I-beams and reinforced concrete. The total weight of iron- and steel-work in the line is 430 tons, of which 61 tons is the weight of the ropes and 260 tons the weight of the structural steel, the remainder being anchor-bolts and rails and the mechanical parts. There are 1,650 cu. m. of masonry, 55 cu. m. of concrete for counter-weights, 40 cu. m. of squared timber, and about 1,500 sq. m. of plank for flooring, roofing, etc.

The rope-way plant was supplied by Messieurs Neyret-Brenier et Cie, of Grenoble, France.

Transportation was, of course, a very considerable item. The total quantity of material handled, including lime, sand, stone, water, etc., is estimated to be about 4,000 tons. The conditions and cost of transportation varied within very wide limits in different sections of the line. Thus, on the mill-end of the line there is a rope-way up to the Pierrefitte mine, from which a wagon-road, practicable during the summer months, leads to the No. 2 angle-station and passes within easy reach of the No. 3 station. This naturally facilitated matters for the lighter materials, such as could be taken up by the present cable. On the other hand, on the Estaing side of the line, material had to be carried 10 miles to the village by carts, and from 2 to 4 miles by men and mules, it being estimated and afterwards proved correct that the point along the line where the cost of transport via the Pierrefitte end was equal to that via Estaing was the intermediate tension-station. As much as possible was carried by mules brought over from Spain for the purpose; but a large proportion had to be carried by men, especially in the winter months.

IV. The Concentrating-Mill.

Mill.—The mill was designed to treat a minimum of 7.5 tons per hr., but in practice the economic average capacity is found to exceed 10 tons per hr. It is composed, as usual, of three sections—the picking-tables, the jigging-plant, and the section for sands and slimes.

The ore as it arrives at the mill is passed over a 40-mm. grizzly, placed directly over a bin of 40 tons capacity to receive the material passing through the grizzly. The oversize goes to the stone-breaker, and thence falls into a small bin by the side of the one beneath the grizzly. The maximum opening of the jaws of the breaker is 45 millimeters.

The ore is fed from each bin into sizing-trommels of 25-mm. round holes, the oversize passing to the picking-belts, and the material below 25 mm. to the jigging-plant. The picking-belts both discharge into a bin of 100 tons capacity, whence the ore is fed into the roller-mill.

The object of this arrangement of the bins is to enable the

stone-breaker and corresponding picking-belt to handle in 10 hr. the quantity of ore put through by the concentrating-machinery in 24 hr., and thus avoid running the cable by night. In this manner also, the quantity of ore to be hand-picked by night is greatly reduced.

There are two additional bins into which the cable-buckets dump the hand-picked lead- and zinc-ore direct from the mine.

The jigging-plant is divided into three units, as follows:

1. For the smalls from the crude-ore and stone-breaker trommels (below 25 millimeters).

2. For the mineral above 25 millimeters.

3. For the rich (lead-zinc) middlings.

The material below 25 mm. passes, without crushing, direct to the first section by means of a launder, that above 25 mm. passes through the 950- by 320-mm. roller-mill before going to the jigs.

The section for slimes and sands is divided into four units, as follows:

A. For the fine sands and slimes proceeding from the first two units of the jigging-plant.

B. For the fine sands and slimes proceeding from the third unit of the jigging-plant.

C. For re-washing the rich middlings from units 1 and 2.

D. For re-washing the poor middlings from units 1 and 2.

In section 1 of the jigging-plant, as before stated, the material below 25 mm. coming from the two trommels which precede the picking-belts is carried by a launder direct to the trommels and *spitzlutten*, and thence to 7 jigs.

In section 2 the ore is fed from the storage-bin into a 950-mm. roller-mill, and thence through trommels and *spitzlutten* to 6 jigs. The oversize (above 12 mm.) returns to the same rollers.

The relative extent of these two sections is, I believe, somewhat unusual in that the respective capacities are nearly equal, but it must be explained that the ore is friable, and that more than one-third as it comes from the stopes is less than 25 mm. in size, and that both in this and in the fines from the stone-breaker there is a considerable quantity of rich mineral.

Section 3 receives all the rich middlings from sections 1 and 2.

A 700- by 300-mm. roller-mill crushes the sizes ranging from 18 to 3 mm., the oversize being returned to the same mill.

The middlings from 3 mm. down are crushed in a No. 3 Heberle ball-mill, the crushed mineral passing, together with that from the roller-mill, to the trommels.

The poor middlings are crushed separately in a 700- by 300-mm. roller-mill and a No. 4 Heberle ball-mill, the roller-mill taking material above and the ball-mill that below 3 mm. The oversize from the roller-mill is returned to the same mill.

The crushed mineral is treated in section 2.

The part of the mill treating sands and slimes is divided up as previously explained.

Section A is composed of 5 Ferraris tables and 2 double Lührig vanners.

Section B is composed of 2 Ferraris tables and 3 double Lührig vanners.

Section C is composed of 2 Ferraris tables and 2 double Lührig vanners.

Section D is composed of 2 Ferraris tables and 1 double Lührig vanner.

Machinery.—The stone-breaker is an old Blake-Marsden, which was taken from the old plant and overhauled. The width of the jaw is 24 in. and it runs at 200 rev. per minute.

The ore is fed into the trommels preceding the picking-belts by adjustable percussion-feeders.

The picking-belts were supplied complete by the Jeffrey Co., of Columbus, Ohio, the belts being made by the Buffalo Weaving Co. They are 50 ft. long and 20 in. wide. The top-carriers are composed of two central horizontal rollers and a hemispherical roller of greater diameter at each side, all on the same spindle, but independent of each other. The speed of the belts is 1 ft. per second.

The roller-mills are from Krupp, of Magdeburg, Germany, and are of their standard type, with grinding-off device attached. This attachment allows the rolls to be trued up in place by means of an emery-wheel, without removing them from the bearings.

The two rollers are driven separately by belts. In the 950-mm. mill the speed of each roller is 41 rev. per min., and in the 700-mm. mill the speed is 58 rev. per minute.

There is a small hopper above the rolls with an adjustable percussion-feed.

The Heberle ball-mills, supplied also by Krupp, have side-discharge and automatic percussion-feed.

The initial charge of No. 3 mill was 60 steel balls 125 mm. in diameter, and 30 balls 100 mm. in diameter.

That of the No. 4 mill was 80 balls 125 mm. and 40 balls 100 mm. in diameter. Wear is compensated by adding one of the larger balls from time to time. Most of the trommels are of steel plate punched with round holes, but the finest screens are of steel wire. They are conical and mounted on horizontal axles.

The jigs were mostly taken from the old plant and overhauled. They are of the usual pointed hutch type and eccentric-driven, the stroke of the eccentrics being adjustable. There are, however, three jigs for the coarser stuff, supplied by Krupp, which are fitted with the crank-arm mechanism. These three, as well as some of the eccentric-driven jigs for the coarser sizes, are fitted with side-discharge.

There is a small hopper beneath each jig to receive the ore run out of the hutch, and beneath the lower jig-platform there is a series of bins to receive the products and middlings. Three tracks run alongside these bins to tram the products to the ore-shed or the middlings to the elevator.

The elevator is on the water-balance system. There are two cages running in a two-compartment tower built of timber. The lower part of each cage is formed by a tank which can be filled with water, and the rope from each cage winds in opposite directions round a drum at the top of the tower, which is geared by means of spur-wheels to a brake-sheave.

The lower part of the cage sinks into a pit below the floor-level, at the bottom of which the water-level is maintained constant by an overflow-pipe. The sides of the tank on the cage project below the bottom, forming a chamber into which the air is compressed by the water when the cage nears the bottom of the pit. This prevents shock on stoppage. The car runs off the cage at the floor above the roller-mill platform, and is dumped into the appropriate hopper.

It was originally intended to use an electric hoist, but the time for delivery asked by the makers was too great. The above scheme, which was adopted, works very satisfactorily.

Section C is provided with a pump for returning to the same

section the rich middlings proceeding from it and from section D, and in the same way section D is also provided with a pump for returning the poor middlings produced therein and in section C.

Close classification is carried out in the sands and slimes group. The Ferraris tables are fed from *spitzlutten*, which give them the quickest-falling particles, and the Lührig vanners are also fed with pulp similarly classified by large *spitzkasten*. The pulp from the large *spitzkasten* is thickened in small *spitzkasten* before going on to the Lührig vanners.

All the shafting is carried in ring-oiler bearings and on the machinery Stauffer lubricators are used where feasible.

The ore-shed is situated alongside the electric railway, from which there is a siding. From the ore-shed to the mill there is a three-rail incline-track 200 m. long and rising at 18°. It is worked on the water-balance system, the trucks consisting of water-tanks with a flat top, on to which the ore-trucks are wheeled from the various platforms of the mill, to which the incline is approximately parallel. At the bottom of the incline the cars are wheeled off the incline-trucks and run into the ore-shed on an overhead track, whence the ore is dumped on to a concrete floor.

The Building.—The building is entirely of timber, the walls being of 1.25-in. tongue-and-groove boarding, with cleats over the joints. It is set on masonry foundations and the terraces are upheld by retaining-walls, also of masonry. The roof is of 1.25-in. tongue-and-groove boarding, covered with "Ruberoid," a thin, tough, water-proof material which stands very well in this climate.

Water-Supply.—The water for the mill is supplied by a stream flowing in an adjacent deep ravine which joins the main river below the mill. Owing to the nature of the ground, it was necessary to place the intake on the opposite bank and considerably higher than the mill. The water is brought some distance in wooden launders, and crosses the ravine in a pipe in the form of an inverted siphon. The pipe is supported by square wooden frames, braced to each other by cross-ties and suspended from two wire-ropes. The timber is so arranged as to provide a foot-bridge across the ravine.

The flow averages about 10 cu. ft. per sec. in ordinary

weather, but may be ten times as much in flood. The mill consumes about 3 cu. ft. per second.

Disposal of Tailings.—In the past the company had had considerable trouble with the farmers on account of pollution of the river-water by the slimes from the mill. The cost of emptying by hand the slime-pits in which the water was more or less clarified was prohibitive, and they were therefore flushed out at intervals into the stream—preferably in flood-time. This, however, gave rise to many complaints, and it became imperative to find a more satisfactory solution. The following scheme was accordingly adopted and works very well:

The jig-tailings are deposited through an iron-lined launder in a ravine about 300 ft. south of the mill in the form of dams, and the slimes are allowed to settle in the pits so formed. The dams were commenced by placing tree-trunks and old sacks filled with jig-tailings across the ravine, from which the water was diverted by means of a flume at a point well above the dams. There are at present two such dams. The first provides a considerable settling-area and the overflow is fairly clear; in fact, it filters through the second dam and rarely overflows, the constant deposition of new jig-tailings providing a fresh filtering-surface as the old surface becomes fouled.

Power.—The power is supplied from a generating-station on the Cauterets river, about 1 km. distant. This station was constructed by a former company with a view to obtaining ultimately, if required, 800 h.p., but, up to the present, one-400 h.p. unit, composed of an Escher Wyss centripetal turbine, direct coupled to a continuous-current dynamo, 400 amperes at 750 volts, has been found sufficient.

The generator as well as the motors was made by Schneider et Cie (Creusot), and the continuous-current system was adopted by the former company largely on account of the proximity of the electric railway, which also runs on continuous current at the same voltage, and can thus supply the necessary power in case of accident at the generating-station or along the line.

The power consumed at the mill is subdivided in the various departments as follows:

	Average h.p.
Roller-mills, ball-mills, and picking-belts, one 100-h.p. motor,	40–60
Rock-breaker, one 28-h.p. motor,	10–20
Four coarse jigs and 5 trommels, one 18-h.p. motor,	17–18
Remainder of jigs and trommels, one 18-h.p. motor,	16–18
Vanner plant, 2 separate 18-h.p. motors,	28–34
Machine-shop, one 5-h.p. motor,	4
Lighting, one 18-h.p. motor running a 120-volt dynamo,	12

The total available is thus 223 h.p., but the actual amount consumed, as read from the ammeter on the mains, is about 140 h.p.

This subdivision was necessitated by the existing motor-plant, all of which was utilized.

Lighting.—The mill is lighted by Cooper Hewitt lamps supplied by the French Westinghouse Co. They run in two series on 110 volts and are of the automatic type, short-tube, 60 cm. in length. Twenty lamps are found to be quite sufficient to light up the whole mill with the addition of 30 incandescent lamps, run in series of five or six, on the main circuit.

Before adopting this light, a trial was made on coarse and fine ore, wet and dry, to see whether the blende, galena, and gangue were easily distinguishable, since the monochromatic nature of the light made this open to doubt. It was found, however, that while ore in lumps required a little practice to distinguish, the fine mineral, especially on the tables, was more sharply defined than in daylight, and in actual practice the results have been very satisfactory. The comparative absence of shadow which characterizes the light thrown by these lamps is a great advantage in a mill of this kind.

The recoveries obtained in the mill average well over 80 per cent. of the zinc- and about 60 per cent. of the lead-contents. The relatively large quantity of blende produced as compared with the lead-content of the ore makes it difficult to secure a higher recovery of the lead without a corresponding loss of zinc, since the blende always contains about 4 per cent. of lead. Probably the above ratio of recovery of lead and zinc represents the maximum economical extraction in this particular case.

The lead-concentrates are dressed to about 67 per cent. of lead, and the zinc-concentrates to about 45 per cent. of zinc. The lead-concentrates average about 28 oz. 1 dwt. of gold per ton.

V. Schedule of Costs.

1. *Rope-way.*

Structural steel, mechanical parts, ropes, and rolling-stock, including erection,	£12,170
Excavations and masonry, including transport of material for same,	4,200
Transport of structural steel, ropes, mechanical parts, timber, etc.,	3,250
Incidental expenses, including accessory buildings, water-supply for hydraulic-brake and cooling gasoline-engine, starting-adjustments, etc.,	1,430
The total cost of the rope-way in running order, including accessory buildings, etc.,	£21,050

The rope-way foundations are constructed of rubble masonry set in mortar, its composition being 1 part of hydraulic lime to 3 parts of sand. The sand was washed from alluvial soil along the cable route, and the water was either obtained from the melting snow or carried in furrows to the proximity of the work. Most of the stone was procured within 150 m. of the spot of erection. Some of the sand, however, had to be carried on pack-mules, in some instances as far as 3 km. up a steep trail.

The lime cost 33.8 francs per metric ton f.o.b. Pierrefitte; the pack-mules, from 8 to 10 francs per day, including muleteers. Masons received 7 francs, and laborers, 4 francs per day of 10 hours.

The average cost of all masonry throughout the rope-way line, including transport, laying and cementing up foundation-bolts and rails, also cement coping and concrete foundations for machinery, was 53 francs per cu. meter.

The total excavation amounted to 8,000 cu. m., and the total masonry to 1,650 cu. meters.

The concrete blocks for the counter-weights were composed of 1 part of cement to 1.5 parts of sand and 2.5 parts of gravel. The cost of cement f.o.b. Pierrefitte was 50.60 francs per metric ton.

The average cost of concrete per cu. m. was 80 francs.

2. *Transportation of Materials.*

The average cost of transporting the structural steel, mechanical parts, ropes, timber, etc., from Pierrefitte station to the spot of erection amounted to about 151 francs per ton, which includes the cost of making about 3 km. of trails for the mules and men.

About 40 per cent. of the material was too heavy to be transported by mules, and therefore had to be carried by men. In some instances as many as 32 men were engaged in carrying one piece of structural steel. The maximum load carried exceptionally by mule consisted of the head-piece of the rope-way standard, which weighed 175 kilograms.

The normal maximum load for convenient packing was 150 kg., but in the great majority of cases, where the pieces were cumbersome, the weights were considerably less. The average load would be about 100 kg.; and the total height to which a mule lifted this in a day would average 1,000 m., the total distance traveled being about 8 miles per day.

The load per man for the cables or structural steel would average about 15 kg., the maximum for short distances being about 25 kg. The height lifted per day averaged about 1,000 m., the distance traveled with the load being about 3 miles.

The total cost of transportation was considerably increased owing to the large quantities of snow which had to be cleared away from the trails, and to the generally rough and broken character of the country.

3. *Mill-Construction.*

The total cost of the milling-plant was £19,400, and in the construction of the mill building and foundations 2,100 cu. m. of rubble masonry were used in retaining- and foundation-walls. The mortar for this masonry was composed of 1 part of hydraulic lime to 3 parts of sand. The stone was quarried within 200 m. of the mill-site, and transported thence by tram-line. The average cost of the masonry per cu. m. was 15 francs. The total amount of masonry for foundations for the various machinery was 340 cu. m., consisting chiefly of masonry set in cement-mortar or of concrete mixed in the following proportions: 1 part of cement, 1.5 of sand, and 2.5 of gravel. The sand was obtained from the old mill jig-tailings, and was transported about 200 m. The water was carried to the mill-site by furrows.

The mill building comprises 300 cu. m. of framing-timber, which cost, in place, 80 francs per cu. m.; 1,800 sq. m. of tongue-and-groove siding, 3 cm. thick, which cost 2.50 francs per sq. m.; 2,400 sq. m. of roofing, 2 cm. thick, covered with three-ply Ruberoid, which cost 6 francs per sq. m. in place.

There are also 1,000 sq. m. of cement flooring, which cost 5 francs per sq. meter.

4. *Mining-Costs.*

Miners earn about 5, laborers 4, blacksmiths and mine-carpenters 6 francs per day of 10 hours.

Dynamite costs 4.10 francs per kg. delivered at the mine.

The average cost of driving headings 1.50 m. by 2 m. works out at about 75 francs per linear m.; stoping at about 4 francs per metric ton. Total mining-costs, including superintendence, development, etc., average about 7 francs per ton of ore delivered at the loading-station.

5. *Costs of Ore-Transportation.*

By the rope-way to the mill, a distance of 7 km., costs on an average about 1 franc per metric ton, not including depreciation, when running day-shift of 10 hr. at full capacity of 10 tons per hr. As the plant has been in operation for only about 4 months, I am unable to give the exact figures for depreciation, wear, and tear. Judging, however, from our experience up to the present, this figure should not be abnormal.

There are four attendants at the two mechanical stations and three at the others, at an average wage of about 4 francs each per day. There are also employed on the line one oiler at 6, one mechanic at 6.50, and two mechanic's assistants at 4 francs per day.

6. *Costs of Milling.*

The total milling-costs average about 2.30 francs per ton, not including depreciation. When working at full capacity of 10 tons per hr., the following men are required for the day-shift:

	Men.	Boys.	Average Wages Per Day. Francs.
Greaser,	1	.....	3.20
Stone-breaker,	2	.....	2.75
Picking-belts,	.....	18	1.65
Picking-belts, foreman,	1	.....	4.00
Crushing-mill attendants,	4	.....	2.66
Jig attendants,	10	.....	2.82
Vanner attendants,	8	.....	2.60
Truckers,	4	.....	3.00
Hoist attendants,	.....	.....	2.87
Dynamo attendants,	1	.....	3.75
Electrician,	1	.....	9.40
Mill-foreman,	1	.....	5.00
Mill-manager,	1	.....	20.00
Ore-shed,	2	.....	2.75

The sacking and loading of concentrates are done by a separate gang of men under contract; the average cost works out about 2.25 centimes per sack of 50 kg., or 0.45 franc per ton.

In addition to the above force, the company also employs mechanics, smiths, etc., in the machine-shops for general repairs, and also a small gang, under an outside foreman, for handling various materials and looking after the tailing-dams and decantation-apparatus.

The total cost per ton of ore, including mining, milling, ropeway, transportation, depreciation, and superintendence, averages about 15 francs per ton.

The local staff, in addition to the manager—Mr. E. C. Hugon, who came out with me as assistant in the designing and erection of the plant—consists of the manager's assistant, Mr. J. H. Hugon, who also acts as surveyor, two assayers, one accountant, one time-keeper, and one store-keeper.

I wish to acknowledge the kind assistance of Mr. E. C. Hugon in the preparation of this paper.

Gold-Dredging on the Choco Rivers, Republic of Colombia, South America.

BY HENRY G. GRANGER, CARTAGENA, COLOMBIA.

(Chattanooga Meeting, October, 1908.)

Introduction.

It still rains in the Choco! I well remember one evening a dozen years ago, while traveling toward the town of Negua in a small *champa* or dug-out canoe, the rain fell so heavily that I had to work my hardest to bail out the water with a large *totuma* or gourd in order to keep the canoe afloat, the native boatman meanwhile plying his paddle in the stern. If any one thinks bailing is easy work, let him scoop out a quart or more at a stroke, 40 times a minute for half an hour, and I venture that he will change his opinion. These small canoes required only one paddler to make the 3 miles an hour which is the standard rate of aquatic progress on the Choco.

Another time, on the Atrato river, about six o'clock in the afternoon, the rain fell so heavily that an empty gourd, 9 in. in diameter and 4.5 deep, was filled to the brim in 90 sec., timed by a watch. Such downpours are typical, but unless caught in one it is not specially noticed. These downpours seldom last more than an hour, and frequently cover only a limited area, occasionally falling on one bank of a river and not on the other. During the night-time, the rain, if gentle, will usually last until dawn, and sometimes an intermittent drizzle will keep up all day as well, although, as a rule, it rarely rains during the day.

The rise of the rivers, due to the rain-fall, varies from a few inches to several feet. One night in August, 1899, the spuds of the dredge *Margaret*, operating on the Atrato river, showed a rise of 16 ft., and for 3 days thereafter occurred the greatest flood known in that district for 32 years. The dredge had all three spuds down, and the 2-yd. shovel dragging on the bottom. A 0.75-in. cable fast to a stout tree, 700 ft. away, stretched tight

as a fiddle-string, with no apparent sag, and sang as every rushing limb brushed below it. The bow of the *Margaret* was cut under and chamfered, and while the free-board was 3.5 ft., the deck at places was almost submerged. Day and night a man stood by the heavy anchor ready to drop it, should the cable part. This experience shows that more than one wire-cable should always be provided for a river-dredge.

Although the rains on the Choco seldom exceed an inch a minute, what must have been the avalanches of water, ages ago, when the great cordillera that separates the valleys of the Atrato and the Cauca thrust its way through the shrinking earth's crust and exposed its backbone of porphyry, granite, and trap, 2 miles above the sea. The hot, moisture-saturated winds from the equatorial Pacific rushed through the dustless sky seeking in vain a chance for condensation until, striking the cold surface of this range, they yielded their burden of water with the force of a thousand monitors, which tore loose the cracked crusts and rolled boulders the size of a house 20 miles from the mother country-rock. In this manner the rocks were crushed and the golden content of the quartz was scattered over what is probably the greatest deposit of auriferous gravel in the world, forming the beds of streams compared to which the present river-systems are but as rivulets.

Until recently the Choco was a part of the Department (or State) of Cauca, the capital of which is Popayan, a beautiful city at 7,000 ft. elevation near the head-waters of the river Cauca. Communication from Quibdo, the Choco metropolis, on forced traveling, formerly occupied two weeks each way. Under the administration of President Reyes, the Choco now has its own local government under the title of *Intendencia*, differing from a Department Government in that it has no Legislature or *Assemblea*, and the chief authority is known as *Intendente*, as distinguished from the Governor of a Department.

There are, as formerly, two Provinces—Atrato and San Juan—comprising the territory respectively of the Atrato and San Juan rivers with their tributaries. The San Juan encroaches upon some of the natural territory of the Atrato in the head-waters of the Quito (including some of its tributaries), which is near the city of Istmina, formerly San Pablo, second only in importance to Quibdo, the capital of both the Atrato and the

FIG. 1.—CHOCO REGION, REPUBLIC OF COLOMBIA, SHOWING RAILROADS BUILT OR BEING CONSTRUCTED.

entire Choco. Istmina is the center of the production of platinum in the Western Hemisphere, and in the office of Don Francisco de B. Carrasco I have seen shipments weighing 100 lb. (Spanish) ready to be sent either to Buenaventura or occasionally to Quibdo for shipment by steamer to Cartagena. These shipments are uninsured, reliance being placed on the honesty of the sturdy negro canoemen, who seldom abuse the trust.

Honesty is the rule in the Choco, but occasionally some one tries to "beat the game," hence the merchants scrutinize each purchase, especially if there is a chance that it has passed through the hands of a *platero* or goldsmith. There are many skillful goldsmiths in the Choco, and not all of them are overscrupulous. If the buyer does not like the appearance of the platinum, he empties some of it out on a clean sheet of paper and rubs it back and forth; if a smudge is left he tells the vendor to "go home and take out the lead."

Before the Russo-Japanese war, at a time when platinum was selling at about $10 an ounce, gilded platinum-dust was occasionally palmed off on the merchants for gold-dust. There is nearly always a stock of "artificial" nuggets cast of pure placer-gold in the hope of the premium of about 20 per cent. above the price of gold-dust that nuggets command for use as stick-pins, watch-charms, necklaces, etc.

Gold- and platinum-washing and rubber-tree planting are the industries of the Choco, but it is only within a few years that cultivated rubber-trees have become a factor in the industrial wealth of the district. Except for a few small shipments of dye-woods (small because of the high freights), and the occasional shipment of railroad-ties, all other industries in the Choco are subordinate to placer-mining.

For centuries, since the earliest expeditions of the Spanish *Conquistadores*, the Choco placers have been famous. In taking out nearly $200,000,000 worth of gold the Spaniards cleaned up all the small creek-bottoms, and the readily worked banks that afforded dump-space and to which water could be brought in ditches. In those early days the platinum was separated and thrown away.

The placers of the Choco are no longer a field for the poor prospector with his rocker and pan. Everything easy was cleaned up by the Spaniards and subsequent *hacienda* slave-

labor. Unquestionably, many of the small streams high in the cordilleras contain gold in paying quantities for ground-sluicing or long-tom work; but the great difficulty of access through an unbroken forest, and the distance from all food supplies, will cause such small isolated creek-work to be left for the time when the opening of quartz-workings establishes a permanent population on the western slope of the range, as it has already done on the eastern slope.

During rainy spells, the negro miner collects water in a small reservoir on top of the river-bank, and does a little ground-sluicing each day with the water secured during the previous night, but the extent of such deposits as have a natural dump seldom, if ever, is enough to justify the installation of a monitor and the necessary water-supply system.

Nearly all the rivers flowing from the west contain numerous bodies of gravel that have a bed-rock either too low to afford a natural dump, or too high and uneven for dredging, although some of these might be worked profitably by hydraulic elevators. Two deposits of this character were equipped with the ordinary hydraulic installation—La Virgen Maria in 1887–89 and Chambaré in 1894, but both had to be subsequently abandoned for lack of dump-space. The values of the gravel, as shown by the small amount of work done, promise good returns if the properties were worked by modern methods.

John W. Pace, of California, with some Colombian partners, including Don Ciceron Angel, equipped the Chambaré deposit with a monitor under about 75 ft. head.

The bed-rock dipped into a point that made sluicing very difficult, and as soon as this was reached Pace sold out and left the company. The pay-streak then showed $2 per cu. yd. for a thickness of more than 4 ft. A year later, under Don Ricardo Ferrer, the river-bank was penetrated a few feet further and the pay-streak showed visible gold. I saw several *batea*-fuls washed, of a size requiring 125 to the cu. yd., which yielded from 1 to 2.5 *castellanos* (from $2.40 to $6) per *batea*. Remains of old mine-timbers were found in the gravel, showing that attempts had been made to work it generations ago, but abandoned, probably, as in the present case, through excess of water.

Beyond a doubt systematic prospecting would reveal a number of deposits amenable to profitable treatment with hydraulic

elevators, although some would have to be operated in connection with a steam pumping-plant, provided there is enough gravel of sufficient average value to insure satisfactory returns. I recommend an expedition with this purpose in view. The ground in many, if not most cases, would be held under private ownership, but a satisfactory working-basis could be readily secured.

The annual output of gold, ranging in value from $300,000 to $500,000, is obtained by small bank- and river-washings when the weather is dry enough to permit. This output could be maintained for centuries, but present prospects indicate that within a few years the annual yield, by modern gold-dredging, will be increased many times.

The Choco as a field for dredging is divided by nature into two divisions, separated by the low water-shed of the Isthmus of San Pablo, about 370 ft. above sea-level. The Atrato, on the north side, with its many tributaries, flows into the Atlantic near the Gulf of Uraba or Darien. The San Juan with its tributaries is on the south side and empties into the Pacific ocean a little north of Buenaventura bay.

The Atrato River System.

The Atrato river, with its source near the town of Carmen at the top of the divide, flows west down the side of the cordillera and then almost due north for the greater part of its course. Its principal gold-bearing tributaries flow in from the west. The lower reaches of the Atrato may be of interest to the gold-dredger if a process be devised that can treat successfully large amounts of very fine and poor gravel, having only exceedingly fine gold visible. This process would necessarily include gold-saving devices to collect all the visible gold and much of the impalpably fine gold that is not visible. Under present conditions, however, dredging on the main river is limited to the stretch between the mouth of the Negua and a point 46 miles to the south and east, where the large size of the boulders would prevent dredging-operations. In most of this area, dredges of the largest capacity would have ample room in the channel, which in places exceeds 200 yd. in width. The gravel is easily dredged, with no more serious obstacle than an occasional snag. The bed-rock is quite soft in the lower portion of

the stretch, becoming more firm toward the source, but up to the farthest point where dredging is possible it is not too hard to be easily scraped by dredge-buckets.

A large portion of this stretch has submerged flats that, after the more easily worked channel has been cleaned out, might be dredged, since the characteristics are practically identical with those of the more prominent gold-dredging fields of the United States.

On the west side of the Atrato, below Quibdo, are extensive flats of a fairly stiff clay which carries very fine gold, occasionally as high in value as $1.50 per cu. yd. This clay overlies the *caliche*, or clayey gravel containing sharp-cornered stones. Both carry gold, the clay being the richer. This material would have to pass through a pump or log-washer before attempting to save the gold, since neither a revolving- nor a shaking-screen could disintegrate the clay.

Below Quibdo the gold is very fine, requiring several hundred colors to make a cent, and the washing is confined to surface-panning. Above the town the gravel and gold gradually become coarser, and in dry weather occasional groups of miners are found working down in sub-aqueous holes, the values increasing with depth. Occasional yields of a *castellano* ($2.40) per day indicate an attractive field for dredging-operations.

The tributaries of the Atrato present various phases to the gold-dredger. Beginning near its source, the first dredgable stream is the Guaitado, entering on the right bank. This small stream has about 5 miles of dredgable course, with some adjoining flats in the lower portion. Washings by the natives indicate fairly high return for a small dredge having 3- or 3.5-ft. buckets.

Next, entering from the left, is the Capa. The main stream for about 15 miles would probably yield returns as good as any of the Choco rivers. A large dredge could be used, but in the upper few miles very heavy gravel, though with no boulders, would be encountered. Native divers claim that the gravel is very rich in certain sections. The Capa has some dredgable flats and three important tributaries. The upper one, the Mumbu, is, by reason of the large size of its gravel, torrential currents, and hard bed-rock, interesting only as a source of power after the fuel now abundant along the banks shall have

been exhausted. The Membrado is a small stream having a few miles of dredgable ground, with occasional small flats, on which only the smallest dredge could be operated. The Tumbutumbudo is an important stream, narrow at its mouth, but broadening out towards its source and showing large deposits of gravel, which, according to the native miners, could be easily worked by a small dredge.

The Andagueda, nearly as large as the Atrato, empties into the latter at the town of Lloro. This river is on the *camino real* or highway to the cities of northern Cauca (now Caldas). It has no important tributaries below the upper reaches, and these are of no interest to the gold-dredger. The main stream maintains its volume with little apparent diminution to a point about 15 miles or more above its mouth, or 3 miles above the town of Bagado, where the bed-rock becomes hard, and the stream blocked in parts with huge boulders. I was in charge of prospecting-work on the upper Andagueda in 1894–95, and made many canoe-trips along its course. Almost invariably I stopped to note the results obtained by the native washers and frequently did some panning myself.

For the first 3 miles the gravel amenable to dredging would average not less than 50 cents per cu. yd. The lower section has a few dredgable flats, but in those of the upper section the bed-rock is frequently too high for dredge-work. The gold-value grows noticeably better up the stream, and approaching the town of Bagado I believe that the gravel would average a *castellano* ($2.40) and certainly a dollar per cu. yd. A dredge for this upper section, in which the stream becomes quite swift in times of flood, would have to have very powerful winches, good cables, and a hull designed to offer little resistance at the bow and no drag at the stern. Practice in New Zealand on streams perhaps quite as swift would afford a good guide.

For 3 miles above Bagado the gravel is remarkably rich. In an exceptionally dry period, about 20 years ago, several individual divers are reported to have extracted a pound of gold a day in some places. In a fairly but not unusually dry spell I have seen results indicating $15 per cu. yd. Facing this stretch is the Chambaré mine, belonging to Ciceron Angel and associates, which is the most attractive elevator-prospect I know of. I have seen the gold in the pay-streak of this mine and

have washed *bateas* of the gravel that indicated several hundred dollars per cu. yd. The success of dredging in this section is problematical on account of the strength of the current during floods, although this obstacle might be overcome by having a dredge specially designed to resist these occasional currents. The character of the bed-rock and gravel is favorable, and it is not impossible to get a dredge there and hold it. Once this is done the probability is that it would work gravel averaging $10 per cu. yd., and possibly somewhat more; certainly, it would prove the most exciting dredging ever done, and the clean-ups would have to be made by absolutely incorruptible men.

The next noteworthy tributary down the Atrato is the Samurindo, a small stream entering from the east, with more gravel in its banks than in its bed. This section, for perhaps 6 or 8 miles, is suitable for small dredges only. The natives wash the gravel on a small scale.

Three miles further down the river is the Tanando, dredgable for 6 or 8 miles or more to the point where it divides into the Motoldo from the right and Jiaco from the left. In these branches, as in the main stream, only the smallest dredges could be worked successfully. The tributaries would show not more than 3 miles of dredgable ground, although there is a considerable amount of gravel at points along the banks. The natives frequently obtain a *castellano* or more per day, and the outlook for a small dredge is quite favorable.

Just above Quibdo from the east is the Cabi, with its tributary, the Purre. A small dredge might be operated above the mouth and also in the Purre. In the upper reaches the natives go to bed-rock by rudely-timbered shafts along the banks, and get at times, in the lower few inches, occasionally as high as a *castellano* to the *batea*.

Opposite Quibdo from the northwest is the Quito, which, however, curves around and has its source in the gravels lying between the San Juan and the Andagueda, as has also its main tributary, the Certegui, which empties in from the left bank about 30 miles above the mouth of the Quito. The Certegui, locally famous as a platinum river, produces gold also. The bed of the stream is wide enough for a medium-sized dredge in the channel, and spreads out in parts to a considerable dis-

tance. Many extensive flats along its course should prove profitable dredging-ground, and occasional sections should yield quite high returns, since a *castellano* a day is by no means an unusual yield for a diver. The Quito is a peculiar stream. So far as I know, no one has ever been to bed-rock in the lower reaches, yet for generations the bars have been and are constantly washed, yielding an abundance of very fine gold for a depth of about 6 in. Below this depth gold shows in every *batea*, but in much less quantity.

The source of the Quito gold is a disputed point. After each flood the bars show as much gold as they had before the last time they were worked. In the rapids of the Quito, especially in times of high-water, indeed in all the streams of the Choco, the canoe traveler can hear the click of the gravel caused by the attrition of the stones in the rushing current, and if he is unfortunate enough to have to disembark, he can feel the gravel rushing by his feet. Does the gold on the Quito bars come down from distant sources, which is indicated by the occasional presence of gold in the *duble* or top earth, especially along the Atrato? Is it brought up and concentrated by the river itself, the lighter sand and gravel moving along? Or does it result from the water-milling of the gravel? In the last-named case the gravel would have to be quite rich. The proportion of quartz and of hard quartz-threaded slate in the Quito is high. I crushed and panned one sample of Quito gravel, and although no free gold was visible, there was a good proportion of sulphides. Unfortunately, the sample was lost before I had an opportunity to test it for gold. A possible venture in the Choco district is a dredge on the Quito, having tables to save very fine gold, and a stacker that discharges on to another hull equipped with a Gates crusher, Cornish rolls, Huntington mills and Frue vanners. There is a chance that it might be profitable to mill the Quito gravel. Large dredges could be operated in the channel of the Quito, but grapples should be provided to handle the abundant snags. The upper reaches show attractive flats along the banks.

From Quibdo to the Gulf of Uraba all the gold-bearing streams come from the Great Cordillera, and enter on the east bank of the Atrato. The first stream is La Troje, having a considerable stretch of gravel, on which a small dredge might do well.

At 12 miles below Quibdo is the mouth of the Negua, meaning "bed of gold," a local name derived from the Indians.

Each of the more important rivers has its *pueblo*, at which there are one or more stores to buy the metal from the miners and to provide supplies; also a tumble-down church, which is visited every few years by a priest who holds a *fiesta*, baptizes the children and marries the adults, frequently attended in the act by their progeny. The *Iglesias* at Quibdo and Istmina, the capitals of the provinces, are creditable edifices and have regular services. The importance of each town is gauged by the quantity of gold or platinum sent to Quibdo or Istmina, and, judged by this standard, Negua is the most important town in the Choco outside of the two capitals, since it is recognized as the principal gold-supply to Quibdo.

Besides the main stream there are the following important tributaries to the Negua: Uppermost, the Naurita, a stream that could be worked by a small dredge for about 6 miles. The natives wash the bed and banks, and a return exceeding in value a *castellano* per day for a diver is by no means uncommon.

Next below is the Nemota, dredgable for 6 or 8 miles with a dredge with 3-ft. buckets, and having adjoining flats which show a promising field. The natives work by ground-sluicing at the head-waters on the high bed-rock, but lower down all the gold is recovered from washings in the bed of the stream or from rude shafts in the edge of the adjoining flats. Wherever there is gravel in these flats it contains sufficient gold to pay for clearing off the barren overlay. The miners search for the gravel by a slender steel-shod pole, which is driven through the overlay and worked up and down until the bottom is reached. The sound at this point indicates the condition: if a dull thud, the bed-rock is barren, but a sharp click shows the presence of gravel. The values in these shafts appear to run about the same, but the gold is in inverse proportion to the thickness of the gravel. I have seen pannings from the stream-bottom near the mouth that showed approximately $3 per cu. yd. before reaching the pay-streak proper.

Opposite the town of Negua is the famous Quebrada Concepcion. Except in excessively wet weather the natives are constantly at work here sinking rudely-timbered shafts along the

banks of this very small stream. The top dirt occasionally shows fine gold, but never in high values. The gravel overlying the bed-rock, although but a few inches thick, is nearly always phenomenally rich, and a single *batea*-ful occasionally yields a return of a *castellano* or more. The stream-bed is insufficient of itself for dredging, but with the adjoining flats the work should prove quite satisfactory.

The principal tributary to the Negua is the Icho, which in turn is fed by the Necoda and the Tutunendo, both workable with a 3-ft. bucket dredge. The latter, judging from the yield obtained by the divers, should prove very profitable. All of these streams have occasional flats that could be dredged. The Icho is frequently abutted by high bed-rock, and consequently in many parts the banks could be worked only by elevators. This stream is nearly as large as the Negua, and could be worked for 12 miles by a large or medium-sized dredge.

All the placer-gold and platinum of the dredgable streams of the Choco is scaly and similar to that of the Oroville district, California. In the Negua it is comparatively coarse and found principally in the pay-streak. However, I have made tests in places on the bars of the upper Negua that showed gold near the surface of a value exceeding $1 per cu. yd. The Negua and Icho, owing to their rich, easily-worked tributaries, are little washed by the natives. But these tributary streams from both sides are so rich in heavy gold, and the character of the gravel in the main stream beds is such, that the gold must be there, probably in still larger quantities. For the past 14 years no one has reached the bed-rock of the Negua, but the late Señor Saldariaga, an intelligent Antioquian miner, showed me a spot where in an exceptionally dry spell he claimed to have sunk to bed-rock at the edge of the stream and from about 2 sq. yd. taken nearly 3 lb. of gold. I do not hesitate to class the upper Negua as an exceptionally attractive gold-dredging section, since all indications point to very satisfactory returns.

Below the Negua is a small stream, the Puné, the source of which is in the flats to the west of the Nemota and north of the Negua. Here a number of the natives are constantly at work on gravel that appears to be dredgable, at least in places.

The river to the north is the Bebarama, a large stream having attractive gravel, with coarse gold and some showing even on

the surface. A large dredge could be used here for many years.

To the north is the Agua Clara, a small stream heading in the extensive flats that have already produced large amounts of gold and platinum. The overburden has much clay, for which special appliances would have to be provided in a dredge suitable for this stream.

The Bebera has its mouth about 60 miles below Quibdo. For the first 3 miles or so of the stream the gravel is so fine and values so doubtful that it is unattractive. Above this point the gold rapidly becomes coarser, and the proved values indicate that it could be dredged at a profit. In slavery times the *pueblo* was an important place, but now it contains only three or four habitable houses, the rest having fallen into decay. The Bebera negro will not work except when it suits him, and that is seldom. His women-folk do a little puny panning, and his small grove of rubber-trees supply his bare necessities. He squats in his "G string" under his *chontadura*-tree and wonders whether it is going to rain to-morrow, passing the time meanwhile in quietly scratching his hide. The settlement of one energetic Andaguedeño, Neguaceño or Condoteño family on the Bebera, with the example of his neat white-washed cottage and well-dressed family (on festal occasions), and the local injection of the phrase in use on some of the other rivers, *El que no tiene plata no es gente* (freely translated meaning, "the man without money cuts no figure") would probably stir ambition among the Bebereños and arouse the mining activity which the gravels of the district justify.

The Arquia, with its mouth a few miles to the north of Bebera, shows only unattractive gravels at the mouth, but from La Isleta, the head of canoe-navigation on the trail to Urrao, for some miles down could be worked by a medium-sized dredge.

The northernmost river which, under present conditions, I consider attractive to the gold-dredger is the Murri, a large stream capable of floating a large dredge, that might be operated profitably from near its mouth for a distance of about 20 miles up its course, both in the river-bed and in occasional flats. The clean-ups, however, should not be estimated too high until the assay-returns are in hand, since the matte or bars would be liable to contain considerable metallic copper from the numerous lodes crossing its head-waters. In some of the tributaries

of the Sucio, having its source near that of the Murri, metallic copper can be obtained by panning.

The Murindo and Sucio both present stretches of dredgable gravel, but the Atrato for the northern 100 miles or more of its length has virtually no ground to its eastern bank, merely an occasional strip of land a few feet wide overflowed at high water and backed by an almost continuous swamp. The tributaries in this section, although sizable streams miles inland, flow into the Atrato through narrow openings, which would render it very costly to install a dredge, either set up or in sections.

The Jiguamiando is another similar stream in this section. These streams will probably eventually be dredged, but not until after the thorough development of the other sections of the Choco. The Sucio, however, will have its broad, gravel-bottomed section made accessible in the near future by the completion of the Colombia Central railroad, which reaches the Sucio near Pavarandocito.

The San Juan River System.

The San Juan river, with the possible exception of the Patia, near the Ecuador line, is the largest stream in Colombia emptying into the Pacific ocean. Occasional panningss have been reported in its lower reaches, but the indications are that it is interesting only from the vicinity of the town of Noanama to the cañon or *angostura*, a stretch of about 58 miles. Dredges of the largest capacity could be used with ordinary river-practice below Istmina (formerly San Pablo). There are many large gravel-flats along the banks below the mouth of the Condoto; the gravel in the river-bed is abundant, with good values. In the lower section of the stretch near Noanama fine gold is found on the surface in greater quantity than in the underlying next few feet. In this locality the pay-streak has never been reached, but as it is so well defined in the channel above, and in the tributary streams, there is little reason to doubt its existence, though of course it will have a correspondingly lower value on account of the greater distance from the source of the gravel.

Below Istmina the bed-rock is sedimentary and soft enough to be fairly well cleaned by buckets, and the values obtained by the washings indicate that profitable results could be obtained.

Above the town the bed-rock is hard and unsuited for scraping by a dredge, but the values are high before reaching the bed-rock, and large clean-ups are almost certain even for a dredge that would neglect the bottom few inches of the gravel. Except on high points along the sides, the bed-rock in the upper section has not been reached for years. I have frequently seen miners digging down in the edge of the stream and getting results exceeding $1.50 per cu. yd. within 4 ft. from the surface, the values increasing with the depth. In this stretch there are many places at which, in dry weather, the divers claim to have obtained results that indicate $5 per cu. yd. During the famous dry spell 20 years ago, there is one spot where it is reported that divers managed to reach bed-rock and obtained single *batea*-fuls that yielded an ounce of gold; and the story is that such was the frenzy of the divers, endeavoring to fill their *bateas*, that one of them was drowned under the press of his down-coming companions.

I once spent three weeks on the upper San Juan waiting for a chance to verify these stories of exceptional richness, but though it was early spring, when dry weather sometimes occurs, the rain fell in the head-waters every night and my tests did not exceed $2 per cu. yard.

Except in part of a small section a few miles above Istmina, where a dike crossed the river, the gravel in the upper San Juan is not too heavy for a hydraulic dredge specially adapted to clean the bed-rock.

The San Juan being a shorter river than the Atrato, and draining a far smaller area, has fewer tributaries, but several of these are noted for their gold- and platinum-bearing gravels.

The river above the cañon runs very swiftly between two smooth rock-walls for several miles, having a boiling right-angled turn at the foot. The gravel of the main river and also of the Pureto would be attractive except for the impracticability of getting a dredge through the cañon, where canoe-travel, even with the most expert canoe-men, is exceedingly dangerous.

Streams that carry gold in quantity always have a swift current, shoals, and rapids. Shooting these rapids in a canoe is an exhilarating pleasure, with very little danger, but going up by tiresome poling is wearisome in the extreme. In order to

paddle across a stream to get on the short side of a bend, the current is utilized by turning the bow of the canoe at an angle of 45° up stream. At times the canoe is forced backwards at a rate of from 8 to 10 miles per hour. On approaching the opposite side of the stream the bow-man drops his paddle and takes his steel-shod pole to check the motion. Generally, all is well, but if the pole breaks or slips, the negro goes headlong into the water, and frequently all hands follow suit. If this occurs at the head of a rapid, a little judgment and swimming will land all hands and the baggage safe to shore in the quiet water below. But in the San Juan cañon there is no shore and a dangerous rapid at the foot. Many of the natives refuse to pass it, and I can testify that its ascent makes a good cigar lose its flavor.

Some miles below the cañon, at the town of Tado, on the southern bank, the Rio Platina empties into the San Juan. The gravels of this small stream and its banks contain little or no gold, but much platinum. A dredge for the Rio Platina would dispense with quicksilver-riffles and depend on matting to save the metal.

Further down is the Condoto, the most noted tributary of the San Juan, and the greatest platinum-producing stream in the world. The Condoto, Platina, Certegui, and Agua Clara yield about 10 per cent. of the world's platinum supply. The largest known platinum-nugget came from the head-waters of the Condoto, and this section of the stream is a most inviting field for systematic prospecting for discovery of the metal in place. To the gold-dredger the first 6 miles only are interesting; beyond this point large boulders and the character of the stream-bed would prevent dredging; hence the following remarks are applicable only to the short section of 6 miles.

On the lower 3 miles, or below the mouth of the Iro, a tributary, of the Condoto, a large dredge could be operated. There is a long supply of high-grade gravel in the river-bed, bars, and islands, and a considerable quantity in the flats adjoining portions of this stretch. Above this section the upper 3 miles calls for a short dredge in order to work the recesses in the bed-rock, which at places appears in irregular bluffs along the stream. The bed-rock of the Condoto looks hard, but is readily cut with a knife, and leaves nothing to be desired for

dredging. I spent more than a week in the Condoto during a fairly dry time, and obtained a number of samples that show values of from 36 cents to $12.69 per cu. yd.; therefore I am prepared to believe stories of the rich finds where in exceptionally dry weather the native divers occasionally reach bed-rock. The values quoted were estimated with a price for crude platinum of a little more than $10 an ounce. There is no place more attractive than the Condoto for dredging, but as about two-thirds of the values are platinum, it cannot be strictly called a gold-dredging field.

The Iro flows into the Condoto at the right bank at about 3 miles from the mouth of the latter entering the San Juan. The Iro has a volume of water as large as the Condoto above this point, and a very much larger area of dredgable river-bottom and adjoining banks. The first 6 miles or so shows a mixture of gold and platinum, the surface-tests indicating a value of approximately $1 per cu. yd. There are also several attractive flats adjoining this section. At the end of this stretch is a short *angostura* or narrows having a bed-rock of a kind of conglomerate that runs across this section of the country; the banks are high, coming close together, so that without clearing away it would be difficult to get any but a small dredge through this stretch; after this point is passed, however, for 10 or 15 miles the stream is broader and the field for dredging is attractive. In dry weather natives can be seen scraping gravel from the bottom at certain places and obtaining very good return for the day's work. There are also numerous flats adjoining the upper Iro, in some of which shafts are sunk, and from the thin pay-streak overlying the bed-rock very high results are occasionally obtained. Coarse gold and even nuggets have been found toward the upper end of these flats. Below the narrows there is much platinum with the gold, which indicates that the gravels were intermixed with the original Condoto supply; above the narrows platinum is not a factor. Owing to the gravels being so spread out above the narrows, the returns from dredging-operations would not be as large as in the Condoto, where the topography confines and concentrates the gold. Nevertheless, the constant yield obtained by natives shows that dredging should be profitable.

About 6 miles below the mouth of the Condoto the Tamana

empties into the San Juan on the same side. The Tamana is a large stream and has extensive gravel-areas in the lower reaches. The conditions are suitable for dredging for about 15 miles up its course. Gold may be found on the surface at the upper end of practically all of its bars. At a point about 6 miles above the town of Novita, or 3 miles beyond the point where dredging could reasonably be expected, there is a famous *zabullidero*, or diving-spot, where in exceptionally dry weather natives are said to have obtained several ounces of gold from the gravel brought up in a single calabash. Surface-gravel from the bottom of the stream-bed at this point showed more than $2 per cu. yard.

Between the Tamana and the Condoto the country is overspread with gravel, much of which is undoubtedly dredgable. A number of square miles of it are so situated as to be available for hydraulic work. A large area of this ground shows a safe average value in platinum and gold of 12 cents per cu. yd. In the bed of the Tamana itself platinum does not occur to any commercial extent.

About 20 miles below the mouth of the Tamana, a short distance above the town of Noanama, the Sipi empties in, also from the southern bank. This river, like the Tamana, appears to have derived much of its gravel from the Cerro de Torra, the lofty peak which is the southern limit of the range that divides the Cauca valley from the Pacific ocean. I have prospected in a part of the Cajon, a tributary of the Sipi. The Cajon shows values in the stream-bottom which would probably permit dredging at a profit. The natives here show more ingenuity in recovering gold from the river-bottom than in any other section of the Choco. They make a *troja* by laying a log across a shoal, staking it firmly to prevent washing away, and then driving a series of poles over this log into the gravel on the upper side at an angle of from 30° to 45°, all fastened with *bejucas*, or bush ropes, made of the flexible vines found everywhere in the tropics. When a freshet occurs the water rushing over the *troja* removes the top stuff and leaves the rich gravels below free to be reached by diving.

All streams running into the Cajon from the Torra carry gold and have principally granite and gneiss gravels. Samples can be easily found that contain sulphides, and all the indica-

tions are that Cerro de Torra is well worth prospecting for quartz-properties. My trip to this stream came about in an interesting way. A few years ago I was consulted by some men in western Pennsylvania who told me that they had an old report showing good milling-values in a conglomerate on a certain stream tributary to the Cajon behind Novita. I told them that I never had tested any of these conglomerates, although I had noticed them throughout that section. They had sent an old miner to investigate these reports, but before his return his reason failed him and he was unable to give an intelligent account of his trip. I accompanied one of the party for a month's test of the ground, and after finding the place we tested a *batea*-ful of ground from the surface and obtained considerable gold. I then selected a certain spot and sweeping away all of the surface conglomerate put down a shot. The material thus loosened was crushed in a hand-mortar, screened and carefully panned, but the result showed a value of less than one cent per ton. A few shots in other places yielded similar results. After the last panning I said: "I will show you how your man deceived himself." I tore up a lot of grass that grew in the crannies of the conglomerate where it was overflowed by every high water, and carefully washed out the roots in a *batea* until I had the *batea* full of sand and fine gravel, which, on being washed, showed enough gold to cover a thumb nail. It was evident that the test on which the hasty report was made did not go far enough through the conglomerate below the grass-roots to give a proper estimate of value.

From repeated stories that I have heard, the main Sipi and various of its tributaries are undoubtedly among the rich gold-bearing streams of the Choco, having large dredgable areas of gravel which, however, will never be dredged for a peculiar reason and one that has prevented me from prospecting them. There is a small fly in this section, the bite of which turns a man's skin blue if he stays there for three months. The skin disease is called *carate azul*, and afflicts all of the inhabitants other than the Cajon. The skin turns a mottled blue, interspersed with white spots, which becomes scaly and itchy, and the odor from a perspiring *caratoso* is quite offensive. It is probable that this little fly serves to transmit the microbe which causes the disease; but this much is certain, that he who goes

there gets it—and I leave the description of the gravels of the Sipi for another chronicler. The disease is said to be cured by a prolonged treatment with mercurial salves that burn off the entire cuticle. I recommend the Sipi river to the investigation of the Medical Association before further prospecting-work is made in that district.

One or two of the small tributaries of the San Juan from its northern shore are said to carry gold, but the information regarding them has not yet been sufficient to induce me to explore them. The Calima, a long stream which one has to travel in order to go via canoe to Buenaventura, shows, for a long distance, gravel said to carry gold. It is possible that the upper end of this stream may prove interesting to the gold-dredger, but I am of the opinion that the gravel was not derived from the gold-bearing country-rock of the main range.

General Remarks.

To a large extent the gravels of all the streams of the Choco are similar, the pay-streak carrying the values being found directly over the bed-rock. As a rule, the pay-streak is of a bluish color, and the natives frequently refer to it as *el azul*. The gravel above the pay-streak carries more or less gold. During freshets the gravel is carried down stream, and while the position of the bars remains the same, it is quite certain that frequently the old gravel is substituted by gravel from above. Hence, where surface-gold is found it is more or less concentrated at the upper end of the bars, where black sand is also found, while the lower end of the bars carries light sand having little or no gold, except where it is extremely fine, as, for instance, in the Quito. About 12 years ago I sank a shaft on the center of a bar below the town of Negua, and at 10 ft. below the surface I found a lot of broken crockery of such a class that it could only have been thrown into the river since the conquest.

The existence of gravel on the bars of the stream does not necessarily indicate ground that could be worked with the present system of gold-dredging. In many cases it may be simply the light, washed gravel that has been cut down from the benches of the ancient deposits and therefore fails to show the characteristics of gravel that has been in place for centuries. That is to say, where the gravel in the adjoining flats is similar

to that of the stream and shows the well-defined characteristics of the gravel of its age, it is gravel in place and is the farthest down-stream point where an underlying pay-streak can be expected.

It is probable that for years to come dredging will be confined to the stream-bottoms, as, owing to the lesser quantity of barren overburden, the average value per cu. yd. is much in excess of that in the adjoining flats, even though the pay-streak of the ancient gravel-flow is as well defined and valuable as that of the river-bottom. This circumstance corresponds to the development of dredging in New Zealand, where before beginning work on the adjoining flats approximately 200 dredges were put to work in the river-bottoms of a much less total mileage of dredgable ground than that presented by the Choco.

Preliminary Testing for Dredging.

The ground is tested for gold-dredging in order to ascertain the value of the gold-content, the character of the bed-rock, the absence of excessive clay, and the size and looseness of the gravel. There are so many miles of river-beds in the Choco in which these factors are so clearly and evidently favorable that it would be waste of time to undertake any systematic drill-tests. Especially so in view of the fact that practically all of the dredgable streams are held under title, by individuals or companies, who will work the ground themselves or lease it on a royalty. Thus, in the Choco it is not necessary to ascertain in advance the reasonableness of high purchase-prices such as have been recently paid for gold-dredging ground in California.

The following preliminary work should be done before deciding upon building a dredge:

(1) Proper design of the hull in order to resist freshets, for which New Zealand practice[1] will be a valuable guide.

(2) The determination of the proper winch which would meet the requirements to resist the strength of the current and admit of an extra length of the working- and warping-lines. A general study of the winches used in California dredging and the data describing the styles used in New Zealand will aid to make a good design. Also consultation with manufacturers making a specialty of winches will be a help.

[1] *New Zealand Government Reports on Mines.*

(3) The proper length of the ladder can be ascertained by using as a basis the depth of water and gravel in the stretch desired to work to bed-rock. This depth is found by sounding in the occasional whirlpools and adding a sufficient length to permit operating in all but the severest floods.

(4) Gold-saving should only be attempted by using the most successful modified New Zealand tables of California practice, and as each new dredge makes some little variation of the general practice, a careful study of this branch of the process will be required.

(5) The style of buckets and tumblers can be determined by a close observation of the valuable data of continuous operation on California practice dredges; and even in spite of the most careful study of the excellent data tabulated by the California State Mining Bureau[2] and by D'Arcy Weatherbe,[3] it is probable that no two engineers would arrive at identical conclusions.

There must be nothing flimsy about a dredge and its equipment for operation so far from home, yet it is by no means easy to determine the relative value of the special features embodied in the construction of the various makers. Until recently the greatest defect of the bucket-lines has been the crushing of the cast-iron bucket-bottoms by stones riding around the lower tumbler. This has been overcome by increasing the weight and using an open-bottom tumbler after the Risdon style. It is possible that just as satisfactory service could be obtained by a much lighter bucket with a cast-steel frame and a riveted boiler-plate bottom. The reduction in weight would amount to several hundred pounds.

I noticed at Oroville that many Risdon bucket-bottoms were dented by stones riding around the tumbler, but none that I recall were put out of service from this cause.

Owing to the almost daily variation of water-level in the Choco streams, all dredges should be provided with a stacker.

For years to come the power for operating the dredges will depend on wood-fuel, and the first dredges should have steam-boilers and engines, although a little later it is probable that better results could be obtained by using modern gas-engines and producer-plants. A plant of this type would save weight

[2] *Bulletin* No. 36 (1905).

[3] *Dredging for Gold in California* (1907).

and space on the dredge and economize fuel more than 50 per cent., but it is not to be considered for use on the Choco until after it has passed the experimental stage and skilled operators are obtainable.

The dredges should be equipped with electric light, which will permit continuous operations during the day and night. There is no trouble in getting natives to work on night-shifts, and foreigners will be found to stand it without detriment to their general health.

In years to come the flats should be tested by Keystone or similar drills, or perhaps preferably with shafts, to verify the conditions and values; but since this class of dredging is far in the future, its consideration can be left for the present. With scores of miles where the record of the *Lady Ranfurly* of 1,200 oz. in a week can be anticipated, the dredging of 15- to 30-cent flats will be left for later consideration.

With regard to the health in the Choco, dredging-expeditions, and in fact all mining-expeditions, should be provided with the proper medicine-chests to combat fevers and stomach troubles, as well as cuts and bruises; but an ounce of prevention is worth a pound of cure, and it will be found that men getting the proper amount of sleep and exercise, bathing daily, avoiding fried food or excessive sweets, and eating freely of oranges and pineapples, and drinking lemonade daily, will have little call for medicine-chests; and when they do feel a little off, 3 or 4 five-grain tablets of Warburg's tincture, taken at night, will correct the indisposition.

I have published a description of the gold-dredging industry in Colombia, The Future Gold Output of Colombia,[4] and will omit any reference to the history, except to emphasize that the outfits formerly used were entirely inadequate for successful gold-dredging in the Choco or elsewhere.

Transporting the Dredge.

One of the most important points in connection with gold-dredging in the Choco is getting the dredge to the ground. There are two means of travel, by the Atrato and the Gulf of Uraba, and by the San Juan and the Pacific ocean. At neither

[4] This volume, p. 315.

of these places are there facilities for erecting a dredge or discharging machinery from deep-sea vessels. Consequently, the hull-material and equipment must be shipped from the nearest port having these facilities, respectively Cartagena and Panama. The shipment in small boats is attended with risk and delay; but the dredge could be completely built and set up (except the bucket-line and gold-saving equipment, which would be stored on deck) either in the United States or at Cartagena or Panama. The dredge that was taken there in the early 80's was built in the Gulf of Uraba with an unsheathed wooden hull, the machinery being loaded from a schooner. The *teredo* bored into the hull while it was being installed, and the dredge sank on its way to the river and became a total loss.

The second dredge was built at Quibdo after great delay in getting material together at the ground. The hull of the next dredge was of steel, and was towed out by the large tug-boat, the *Mary E. Luckenbach*, from Philadelphia to the mouth of the Atrato. In its wake was a small tug to act as a rudder. I made the trip on the small tug and can testify to the practicability of this method of getting the dredge to the ground, provided the hull is absolutely water-tight and equipped with diaphragm bilge-pumps, and in addition has a sufficient number of fenders to break the force of the waves coming over the bow. The towing-bill for the trip was a little more than $9,000. The dredge left Philadelphia May 25, 1899, and was at work digging gravel by August 7. It required about 24 days to tow the dredge from Philadelphia to the Gulf, and 16 days and nights to tow it up the Atrato. Two weeks were spent in the lower Atrato fitting up the boom, spuds, and sluices, as well as building living-quarters on the upper deck. The time consumed in towing from Philadelphia would certainly have been much less if the side spud-frames had not been erected before leaving. It is interesting to know that the hull of this dredge is perfectly sound to this day, and will possibly be remodeled by putting a well into it and converting it into a modern dredge.

The third dredge, erected at Cartagena, had the hull built of wood sheathed with zinc. It required several months to build and launch the dredge, and trouble was had later in setting the machinery, since it was necessary to depend on local steamers that could spare the time to transfer the machinery from the

wharf to the hull. When it was finally ready there was several months' delay in getting a steamer to tow it to the mouth of the Atrato and to insure it while *en route*. More months were consumed in getting it up to Quibdo and ready to operate, so that even excluding the interest on the investment, it cost far more to get this dredge to the ground than the previous one which was towed from Philadelphia.

A dredge now being built under the design of the New York Engineering Co. for work on the Certegui has its material *en route* all the way from New York to Quibdo, with many months consumed in the shipping and no telling when the installation will be finished.

Until the completion of the Colombia Central railroad docks across the Gulf of Uraba and the erection of proper facilities at the mouth of the San Juan, I consider the best way of getting dredges to the Choco is to tow direct from the United States to the Atrato, or from Panama on the Pacific side; the dredges to be completely set up, except the bucket-line, ladder, stacker, and tail-sluices.

Conclusion.

While this paper is written not as an argument for investment but simply as a record of a potentially very important gold-dredging field, the location of this field in a Spanish-American country suggests a few words on the political aspect and the attendant financial phase of the country. I have lived in South America, nearly all of the time in Colombia, for more than 14 years, and can testify to the gross misconception of the average American regarding Central and South America. One hears a lot of nonsense about the instability of Spanish-American governments, revolutions, etc. The severest revolution that Colombia ever had, and which will almost certainly prove to be her final one, lasted from the fall of 1899 to the spring of 1902. From personal observation I can bear witness that real property owned by the natives was respected. Titles to mines and ranches were not only unaffected but could be validly transferred only under circumstances that conclusively proved coercion was not a factor. Foreign property and persons may be said to have been absolutely sacred, and I am unaware of any foreign enterprise in the Republic of Colombia having been adversely affected by this revolution, and I never heard of foreign property rights

or franchises that have been vitiated in Colombia during or by reason of any changes of government. Further, I believe that this is not only true of Colombia but of all Latin-American countries where the foreigner keeps out of politics, and I believe that little would ever have been said on this score if it were not for the fact that in an adjoining country a group of foreigners who financed and instigated a revolution, when they failed were not square enough to stand their just assessment but sulked behind their would-be rights as foreigners and attempted to embroil the United States in another act of paramount injustice.

Capital in the United States, with the kaleidoscopic changes of executive sentiment, sweeping revisions of existing laws, and unpunished outrages of anarchistic labor, has no advantages whatsoever over that in Latin America. The unjust and unreasonable prejudice against the latter country has existed probably as much from ignorance as from any other cause.

The investment of hundreds of millions of American capital in Mexico has brought to the front the fact that while the great opportunities of the wonderfully rich continent of South America have as yet attracted but few millions of American money, the English, according to Charles M. Pepper, the American who best knows South America, have over two billion dollars invested in the continent. Colombia, with its great natural resources, its good and just laws, and its wise administration under President Reyes, is attracting a great deal of attention, largely through its proximity to the Panama canal, which will serve the interests of both its coasts.

From present indications I feel that Colombia will be the field of the world's most active development in the next decade, especially in regard to mining. One modern dredge is now being erected in the Choco, four more are to be ordered, and as soon as the magnet of success asserts its attraction, the Choco region of Colombia will rapidly develop into a very important gold-dredging region.

Table I. is a record of a few samples of gravel which does justice only to the Condoto. With the same attention and with equally favorable weather several of the other streams should show results quite as favorable. This record is not presented with the intimation that gold-dredging would average in all

cases anything like the values given, but simply as of interest in connection with the subject.

TABLE I.—*Gold in Colombian Placers.*

No. of Sample.	Place.	Quantity Treated. Cu. Yd.	Weight. Grains, Troy.	Approximate Value. Cu. Yd.
1.	Bank of Rio San Juan, near house of Santos Torre, above San Pablo,	2	13.158	$0.19
2.	Upper Condoto,	1	107.195	2.67
3.	Upper Condoto, 1 *batea*,	...	1.32	1.55
4.	Upper Condoto,	$\frac{1}{3}$	153.795	11.52
5.	Upper Condoto,	$\frac{1}{8}$	31.27	6.24
6.	Upper Condoto, 3 hours' work of one woman, about	$\frac{1}{4}$	74.6	7.44
7.	Middle Condoto,	$\frac{1}{12}$ (top)	6.22	1.80
8.	Middle Condoto, bank of river, 100 yd. from center of channel, from bed-rock blow-up,	1	40.185	1.00
9.	Middle Condoto, top scraping,	$\frac{1}{4}$	8.435	0.77
10.	Middle Condoto, 1 cu. yd. from top below 3 ft. of water, 75 yd. from channel, no sign of bed-rock,	...	46.615	1.16
11.	Lower Condoto, Playa head Brazo Salvador, 25 *bateas* from *cuerga*,	$\frac{1}{5}$	22.5	3.25
12.	Lower Condoto, Playa Cyprian,	1	32.95	0.82
13.	Lower Condoto, island above mouth,	$\frac{2}{5}$	7.75	0.56
14.	Lower Condoto, of surface,	$\frac{2}{27}$	2.95	0.94
15.	Lower Condoto, Brazo de Salvador, of surface,	$\frac{1}{2}$	8.41	0.36
16.	Lower Condoto (washed by child),	$\frac{1}{18}$	2.425	1.26
17.	Middle Condoto, Playa Mandinga,	$\frac{1}{27}$	18.885	12.69
18.	Tamana, opposite Novita, sample of surface-gold; high water prevented reading away from top edge,	...	1.78	.....
20.	Tamana-Tigre; high water prevented test of value. Top under water, 4 ft.,	$\frac{3}{54}$	4.265	2.16
21.	Lower San Juan, Playa de Mico, end No. 6, S. J.,	$\frac{1}{2}$	6.125	0.40
22.	Lower San Juan, Playa del Mico, surface,	1	14.66	0.36
24.	San Juan, above mouth Tamana, top to 2 ft. deep,	1	15.635	0.51
25.	San Juan, below mouth Condoto, $3\frac{2}{3}$ cu. ft. equals	$\frac{1}{27}$	1.235	0.81
26.	San Juan, Playa midway between Tamana and Mico,	$\frac{1}{5}$	3.805	0.55
27.	San Juan, below mouth Tamana,	$\frac{1}{2}$	5.05	0.24
28.	San Juan, Carampaima,	$\frac{1}{4}$	6.585	0.64
29.	Tamana, head Brazo Eustacio,	$\frac{1}{9}$	1.08	0.24
31.	Atrato, Brazo de Samurindo,	$\frac{1}{7}$	4.54	0.91
32.	Negua, Bocado Nemota,	$\frac{1}{4}$	8.98	1.32
35.	Cabi, Vuelta de la Isla,	$\frac{1}{3}$	6.85	0.78
36.	Cabi, Playa de Paila,	$\frac{1}{3}$	4.575	0.45
37.	Cabi, Cabecera del desecho,	$\frac{1}{6}$	6.595	0.54
38.	Sample showing concentrates with very fine gold,	...	.....	.....
40.	Return of $1\frac{1}{4}$ cu. yd. washings near bank of Atrato, below Quibdo,	...	.....	1.50

The Kaffir Mine-Laborer.

BY THOMAS LANE CARTER, JOHANNESBURG, SOUTH AFRICA.

(Chattanooga Meeting, October, 1908.)

THE history of mining in South Africa differs somewhat from that of other countries in the part taken by the aborigines in the development of the mineral deposits. The Spaniards in America, and the fortune-seekers of Australia and New Zealand, found large numbers of natives in these new countries, but the inhabitants did not become willing or efficient workers in the mines which the white men started.

The Kaffirs of South Africa have not shown the same antipathy to labor, nor do they show any signs of dying out in the presence of the white man's civilization. They are a virile race, and so far the vices of civilization have not killed them.

While it is true that the brain and skill necessary for the exploitation of the mineral resources of this sub-continent have been supplied by the white race, it is also a fact that most of the rough labor in the development of the mining industry of South Africa has been done by the Kaffirs.

The Kaffir tribes differ greatly in their mental and physical capabilities. Compare, for instance, a Zulu of Natal with a Bechuana, or a Xosa of Cape Colony with some of the degenerate tribes of Central or East Africa, and you will find that they have little in common, save that their skin is black and that they do not believe in clothes. As a general rule, the tribes in the milder, healthier climates are far superior, both in mind and body, to the Kaffirs brought up in the hot malarial sections of Central Africa; and it naturally follows that the black men from the temperate zones of the country are much finer workmen than the degenerates from less favorable sections.

It is universal in South Africa to speak of the Kaffirs as "boys." No matter if he is 90 years old, the Kaffir is always a "boy," and in this paper the word "boy" is synonymous with Kaffir.

At the mines there is a great difference in the popularity of the various tribes with the white workmen. Every one prefers working with the Shangaan Kaffirs from Portuguese East Africa, who have been employed intermittently at the mines for many years, and are quite fair workmen. They are civil, obedient, fairly intelligent, and give no trouble, but their failing is a love for alcoholic beverages.

The Xosas from Cape Colony are by no means so popular on the Rand, and some managers refuse to take them as long as there are other natives available. Frequently they are insolent, disobedient, and self-willed, and unless the white man knows them well and can speak their language they give trouble.

With the right kind of white man the Xosa does good work. Physically, he is a superior class of man, but he requires more careful handling than the others.

The natives from Mozambique and Central Africa are unpopular with the miners, who declare that they see little difference between the natives from these parts and an equal number of monkeys. They certainly are stupid, and are a most trying lot to break in. They are very docile, however, and when they learn the work are satisfactory. As a rule, they work for longer terms than the other natives.

The Zulu makes a fine mine-laborer, but there are not a great many of this tribe at work on the mines of the Rand at present. The Basuto boys are good workers, but they are very independent, and do not stay for many months.

Table I. gives the resident population of British South Africa of natives, whites, and colored.

TABLE I.—*Resident Population of British South Africa*, 1903.

Territory.	Natives.	Whites.	Colored.
Cape Colony,	1,424,787	579,741	405,276
Natal and Zululand,	904,041	97,109	107,609
Transvaal (including Swaziland),	896,284	300,225	23,946
Orange River Colony,	235,466	143,419	6,160
Southern Rhodesia,	570,830	12,623	1,944
Basutoland,	347,731	895	212
Bechuanaland Protectorate,	119,411	1,004	361
Total,	4,498,550	1,135,016	545,508

Apart from the native population of the Transvaal there are a large number of temporary residents who are not included

in Table I. This number varies from time to time, being approximately 200,000 at present. The statistics in Table I. for colored (Asiatics) do not include indentured Chinese. In addition to the native population of Southern Rhodesia there are, at present, about 20,000 outside native laborers at work on the mines.

South of the Zambesi river, in British South Africa, about 4,498,550 Kaffirs reside permanently, while the white population is about 1,135,016, an approximate ratio of 4 blacks to 1 white.

The institution of the dual system of labor in South Africa commenced with the introduction of slavery. As was the case in the Southern United States of America, the white man considered it was his mission in life to do the supervising, while the black man did the work. Slavery in South Africa was abolished in 1834, and since that time the white man has done some manual labor, but the country still rests on the dual system of labor. In the phraseology of the country, the white man performs "skilled" labor, while the work of the Kaffir is referred to as "unskilled."

There is no law to define skilled and unskilled labor, and in all classes of work the Kaffir can participate. The white man is not protected against the competition of the Kaffir by any such ordinance as the Labour Importation Ordinance of the Transvaal, an enactment which guarded the whites from any competition of the imported Chinese in the sphere of skilled labor. So far as the law is concerned, there is no reason why the mines should not employ black men in nearly every department, from manager to office-boy. The Kaffir capable of being a skilled carpenter on the mines is only kept from this post by the force of public opinion.

The spheres of work of the two races are well defined and fairly well recognized, and public opinion is so strong that the Kaffir seldom or never tries to leave his sphere. An American negro once came to the Rand with the intention of following his profession of mine-surveyor, but he soon found that he had struck the wrong place, and he then left for another field.

Another reason why the white man has had so little competition from the black man in the skilled lines of work is because of the lack of energy and ambition on the part of the

Kaffir, who is a spasmodic worker and never remains long enough at any class of work to become expert. The ideals of the two races are utterly different. One of the principal aims of the average Kaffir is to get through life by doing the minimum amount of work, and to have sufficient wives to labor for him. Under the old Zulu dominion the sphere of a man was fighting, not working, and women only were regarded as meant to labor. This antipathy to work dies out very slowly.

Moreover, the wants of the Kaffir are rather meager. The white man's bare necessities are the Kaffir's abundance. With the comparatively high wages he earns at the mines, the uncivilized Kaffir can amass sufficient wealth in a year or so to last him for his lifetime.

The British government has dealt far too gently with the Kaffir races. The Boers have often gone to the other extreme, but they have a better idea of the management of these semi-savage people than have the British. Everything has been made easy for the Kaffir. Vast stretches of land have been open to him and his flocks, and he can get a competence with very little effort. The burden of taxation is scarcely felt by the Kaffir, and the necessity for him to work continually has been lacking.

Many now declare that the natives must be compelled to do their part in the development and progress of the country of which they are subjects, or else they must be elbowed out. They must be treated justly but firmly. Under his own government the native is subjected to severe discipline. Soft-hearted people declare that the native must be "persuaded" or "encouraged" to work, but for more than 50 years the blacks have refused to yield to moral suasion, especially in Natal, where from 60,000 to 70,000 whites were formerly forced to import labor from India, notwithstanding the fact that there was a population of 750,000 Kaffirs in that colony.

Down to the present time, therefore, the skilled mechanic or carpenter or miner has had little to fear from the competition of men who will not remain long enough to learn thoroughly any trade or work. But in the sphere of unskilled labor the Kaffir has had, and still practically maintains, a monopoly. South Africa does not possess that class of white men known as "navvies" in England. Heretofore the white man has found

it impossible to compete with the Kaffir for the rough labor of the country, on a strictly economical basis, without sinking to the level of the Kaffir. All the conditions are against the white man performing this labor even if he so desired, for the burdens of taxation and responsibility rest almost entirely on his shoulders, and he is handicapped from the start in any race for the unskilled labor of the country. While there are Kaffirs to "hew wood and draw water" the wages of the white laborer doing the same work will be low.

Owing to the stress of circumstances and the destitution of hundreds of white men, the mines, municipalities, and governments of South Africa have given these men relief-work, but these measures have always been considered temporary. In a few lines of labor requiring skill, the efficient white man is as cheap even at a higher wage, but where brute force is required very few white men individually will do more work than a good Kaffir.

Some people see a change coming over the Kaffirs of South Africa. Their old occupation of war and bloodshed is now gone, and they are forced to be peaceful. They are increasing and gradually becoming civilized. Their economic wants will gradually force them from the kraals, in larger numbers, and send them to the mines. If he can be induced to remain permanently at mine-work, will not the Kaffir leave his sphere of unskilled labor and try to enter the white man's domain? Labor leaders who have visited South Africa freely predict that this will happen in the future, and that the white artisan in the Transvaal, who is now one of the highest paid workmen in the world, cannot hope to hold his position indefinitely. For fear that the wage of the white man will be brought down to the level of the Kaffir in the future, one labor leader, a well-known socialist, has boldly made the rather impossible proposition that the pay of the Kaffir be brought up to the standard of the white man without delay.

In the Transvaal, the practical working of the Chinese Labour Importation Ordinance makes it impossible for the Chinaman to enter the sphere of "skilled" labor. Should the competition of Kaffirs grow too keen, it is possible that, in the Transvaal at least, legislation will be enacted to protect the white man. This idea is foreshadowed on a small scale in the present

mining-laws of the Transvaal. For instance, it is illegal for a Kaffir to do any blasting in a mine. He can assist a white man in his work, but the actual charging-up must be done by the white man. The Kaffir is not allowed to handle explosives on his own responsibility. This regulation is more for the protection of the labor of the white man, since experience in other parts of the world has shown that intelligent negroes can conduct blasting-operations as well as most white men. No Kaffir can be in charge of a man-hoisting engine in the Transvaal; neither is he allowed to give signals to the engine-driver for the raising and lowering of people. There are several other positions on the mines which by law must be filled by white men.

Were there only the economic view in the running of the Transvaal mines to be considered a great deal more use could be made of the Kaffir than at present. Those who have studied the Kaffir declare that if he were given enough attention he could be taught to perform much of the skilled work now done by the white man; and in time, it would probably be possible to run the mines of South Africa with only a few white men as heads of departments, the actual work being done by Kaffirs.

But the economic view is not the only one. The statesmen have a great deal to say in the matter, and they all have before them the ideal of a partly white South Africa. It is their desire to see an increase in the white population of this country, and while most of them recognize the impossibility of employing all whites on the mines at present, it is certain that they will not allow economic laws to drive the white man out of South Africa. The future will witness a struggle for the labor between the white man and the Kaffir, and how it will end no man can foretell.

The environment in which most of the Kaffir mine-workers are brought up is not calculated to make good laborers of them. The burden of the work falls on the women, while the men of the tribe bask in the sun or travel from kraal to kraal. The Kaffir is distinctly a sensualist, and as one authority has said: "His belly is his conscience." The life of the kraal is on a low plane, and save that the Kaffirs do not now go out on fighting-expeditions, the majority of them differ little from those with whom Europeans first came into contact.

Agriculture is followed in the most primitive form, the principal crop grown being mealies (Indian corn or maize). The Kaffirs are still a pastoral people to a great extent, and delight in large herds of sheep and cattle.

Most of the natives now wear a piece of calico about their loins, but the use of skins of wild beasts and domestic animals is still quite common. Some tribes look upon clothing as a superfluity. I have seen hundreds of Kaffirs in Central Africa clothed with only a smile.

Were it not for the outside pressure, few of the Kaffirs would ever leave their kraals to seek work at the mines. Throughout South Africa the Kaffirs are compelled to pay a hut tax, usually $4.85 per hut per annum, and to get this money many of the young men and boys are forced to seek work at the mines.

The labor-agents also induce thousands of Kaffirs to come out to work, sometimes by advancing them money with which to purchase cattle. This money is paid back month by month by the natives. The agents make it a point to stand in well with the chiefs of the native villages by giving them presents, and, as a rule, the most prized gift is a jug of whiskey or brandy. It is by the orders of the chiefs and headmen that the Kaffirs come to the mines.

The natives of South Africa are not allowed to roam indiscriminately over the country. The government of the Transvaal has a firm hold over them through the Pass law, under which it is not legal for a Kaffir to leave his district without obtaining a pass signed by the magistrate. There is no difficulty in getting a pass, but if he attempts to travel without one he is arrested and punished, by either a fine or imprisonment. In this way a check is kept on the movements of the Kaffirs.

Every Kaffir must be in possession of an identification passport, by which the holder can be identified. If an officer of the law finds that a Kaffir is not in possession of a passport he arrests him. The Kaffir is also required to have a small, or monthly, pass. The employer pays for these passports, and is responsible for the taking out of the monthly pass and its safekeeping. The fee charged by the government is $0.48 per Kaffir per month.

Should a Kaffir at a mine wish to leave the property or

travel on a railroad he must obtain a special pass from his boss, which is given without charge. The Kaffir does most of his visiting at the week-ends. The permit to travel is specific. If a native is found wandering without a proper pass, or traveling otherwise than in the direction indicated by his pass, he may be arrested by an authorized officer. This control prevents idling and vagrancy.

When a labor agent arrives at a mine with a gang of Kaffirs they are taken to the nearest pass office, registered, and a passport obtained for each native. The pass officer makes certain that the natives are under no misapprehension in regard to the terms of their contract. If natives come to a labor agent independently they must register and obtain a passport within 24 hr. Any one employing more than 20 natives must keep a register containing full details.

Natives are entitled to demand their passports only on the expiration of their contracts. Should one desert before his contract has expired, he is punished when caught with a $50 fine, or imprisonment not exceeding three months.

In every native district there are European native-inspectors, appointed by the Governor, whose duty it is to hear the grievances of the Kaffirs and to decide the punishment for minor offenses. These inspectors are of considerable help to the mines, for if it were necessary to run to the courts with every small offense of a Kaffir much time would be wasted. For serious offenses the culprit is taken before a magistrate. Except by order of a competent court and of the inspector, the mines cannot make any deductions from the pay of Kaffirs. The inspector has a fixed day or days per week to visit each mine.

The recruiting of Kaffir labor is quite a business. The majority of the Kaffirs must be found and brought to work, only a small percentage coming to the mines voluntarily. In prewar days the mine-labor was delivered to the companies by individual recruiters, who received a fixed amount for each Kaffir supplied. The nefarious practice of "touting" was quite common. A recruiter would supply a mine with Kaffirs, and after a few months he would bribe them to run away from the mine. He would then arrange the passes so as to supply these Kaffirs to another company for a large sum. Formerly there was a large traffic in Kaffir deserters, and in order to stop it practi-

cally all the mining companies decided to combine their efforts in procuring Kaffir labor. In this way the Witwatersraud Native Labour Association was formed, and until recently the recruiting of native labor was almost entirely in the hands of this organization. An outcry against what was designated a monopoly was raised, and people joined in denouncing this association, forgetting the evils that it had corrected. Permission to recruit labor for the mines was then given to any licensed person or persons, and, as a result, there is now practically free trade in the supplying of Kaffir labor to the mines in all the districts of British South Africa. The Witwatersrand Native Labour Association still maintains its position, and to-day supplies about 74 per cent. of the native labor.

Of late the Transvaal government has taken great interest in the question of Kaffir labor. At the head of the Native Labour Bureau is a Director, who has numerous officials under him. The Government Native Labour Bureau has a compound through which all natives recruited for the mines in any district of British South Africa have to pass. Here they are medically examined by government officers to see if they are fit for work, and their contracts and passports are issued to them. It is merely a registering office and does not at present recruit in any way, though it is said that arrangements have been made with the authorities in the Cape Colony under which magistrates can send up natives for work direct to the Government Labour Bureau without the Kaffirs passing through the hands of recruiters. Up to the present time this arrangement has hardly been brought into force. The natives who come through this compound, as already stated, are Kaffirs recruited from British South Africa to work at the mines. Natives who come for work in towns, or who come at their own expense for work at the mines, do not pass through the bureau, nor do natives from Portuguese Africa.

The cost of getting native labor to the mines varies considerably from time to time. When the crops fail there is generally a plentiful supply of Kaffirs, many of whom walk to the mines seeking work. When Kaffirs come to the mines looking for work they can be employed by those mines represented in the Witwatersrand Native Labour Association on the payment of $0.65 per head. Just now the price paid to recruiters for

Kaffirs who sign for six months is about $11 each; for nine months, about $20 each; and for a year, $25.

A Kaffir cannot sign for more than a year at a time, but at the end of this period he may sign again. The total cost for recruiting Kaffir labor is spread over the period of service, a proportional amount being charged out to work-costs every month.

The amount paid per shift to the Kaffir depends upon the class of work to which he is put. Whenever possible he is given contract-work, but he is not keen for it. Unlike the Chinaman, the Kaffir prefers to do a medium amount of work and earn a small wage, rather than exert himself and earn twice the pay in one day for double the work. Every effort is being made by mine-managers to increase the efficiency of the Kaffirs. On hand-drilling the natives are on contract at 1 cent per in., with a bonus over a certain footage drilled per shift. On this contract the expert Chinese driller earns from $22.50 to $25 per month, but so far the prospect of earning this high pay has not induced a large number of Kaffirs to get out of their slow, steady stride. Perseverance will no doubt tell in time, and managers are confident that they will yet get more work from the Kaffirs who work on hand-drilling. There is practically no double hand-hammer work in the mines; it is all single hand-drilling. On machine work he must drill four holes 6 ft. deep to earn his "day's pay." Many mines are now giving a bonus of 4 cents per hole for every hole drilled above four.

In shoveling and tramming on contract the Kaffir is paid at a fixed rate per truck. Table I. gives the latest schedule of the Chamber of Mines for the different kinds of work.

At the present time only about 11 per cent. of the total Kaffir labor employed at the mines of the Transvaal comes from this colony. Of the 89 per cent. brought from outside districts of South Africa, 60 per cent. comes from sections outside of the British Empire.

In view of the fact that the recruiting of Kaffir labor is so expensive, and that the Kaffir only remains at work for periods varying from 6 to 12 months, the idea of moving the Kaffir communities nearer the mines has been often discussed. The Kaffir would only be brought to these reservations on the distinct understanding that he work at the mines for 8 or 9 months every year.

TABLE I.—*Chamber of Mines Schedule of Prices of Labor Per Shift.*

(1) *Piecework.*—Hand-drilling underground, 1 c. per in. drilled (this rate to apply to hard rock only, the rates for softer rock to be fixed by the mine-managers).

(2) *Daily Pay:*

Mine.	
Machine helpers,	$0.40
Hammer boys,	0.36
Shovel boys,	0.30
Tram boys (10 cu. ft. trucks),	0.28
Tram boys (16 cu. ft. trucks),	0.36
Timber boys,	0.28
Stope-gangers and assistants,	0.48
Station boys (where white man employed),	0.28
Station boys (where no white man employed),	0.60
Air-hoist drivers,	0.48
Dry-shaft and winze boys,	0.40
Wet-shaft boys,	0.48
Wet-shaft (developing mines),	0.60
Boys cutting hitches for timber,	0.36
Surface.	
Stokers (12 hr.),	0.60
Stokers (8 hr.),	0.40
Engine-cleaners,	0.36
Sorting boys,	0.48
Head-gear boys (where white man employed),	0.32
Head-gear boys (where no white man employed),	0.60
Mill.	
Elevator boys,	0.48
Vanner boys,	0.48
Mill boys (12 hr.),	$0.48
Mill boys (8 hr.),	0.34
Blanket and sluice boys,	0.48
Crusher boys,	0.32
Surface trammers,	0.42
Mule-drivers,	0.60
Pumpman's assistants,	0.40
Plate-layer's assistants,	0.36
Pipeman's assistants,	0.36
Cyanide Works.	
Solution-shed boys,	0.32
Boys filling and discharging,	0.42
Zinc-cutters,	0.36
Tramming residues,	0.32
General.	
Fitter's-boys,	0.36
Blacksmith's boys (strikers),	0.60
Blacksmith's boys (helpers),	0.32
Carpenter's boys,	0.28
Mason's boys,	0.32
Police boys,	0.60
Compound cooks,	0.48
Drill-packers,	0.24
Drill-sorters,	0.36
Surface laborers,	0.28
Office and store boys,	0.60
Assay-office boys,	0.60
Coal boys (off loading),	0.36

At first sight there is much to recommend the idea of having large Kaffir reservations near the mines of the Rand, but on investigation it is seen that there are many objections.

In the first place, it must be remembered that lines of communication are now so well-established in South Africa that Kaffirs living at a great distance from the mines can reach their homes in a few days' journey. If they were intent on more continuous labor at the mines, the Kaffirs could have several weeks' or months' holiday every year, and then return to work. Except at mines favorably situated, the native women and chil-

dren could not live at the mines, but would have to live several miles distant, so that the Kaffir workmen would be with their families only at week-ends.

But a more serious objection is the economic one. It has been pointed out by Howard Pim,[1] that one reason why the Kaffir is able to work for such a small wage is the fact that his food is obtained for nothing in the kraal. The women and children do most of the agricultural work in the native villages, so that the men have no burden to bear in the support of their families, and can therefore afford to work for very little, because they can save all their own earnings.

The majority of natives would object to being packed into small places near the mines, for they would be in confined quarters and in surroundings among which their native customs could have no place. They would also be forced to buy from white men food which to-day is grown by their women and children. At the present time South Africa could not supply this demand, and more food-stuffs would have to be imported from over the sea, which would mean a great increase in the living-expenses of the Kaffirs, and of necessity a higher wage would have to be given to them.

Then what would become of the women in these congested areas, in which it would be impossible for them to grow their crops as they do now in their kraals? It is illegal to employ females in mining-work, so no employment could be found for the majority of them, and under these conditions they would become demoralized and degenerate rapidly.

Another objection to the crowding of natives into reserves is the fact that, when they become old and infirm, or get disabled or ill, they must be supported. From whence is their support to come? At the present time these "worn out" Kaffirs retire to their kraals, which are more or less health resorts, and get well or else end their days in peace. Moreover, the tribal system of the Kaffirs keeps them under discipline, and if they become criminals, no trouble or expense is experienced by the white race in dealing with the culprits. On the other hand, the expense of policing these areas near the mines would be considerable, and would fall on the white population. Ameri-

[1] Some Aspects of the South African Native Problem, *British Association for the Advancement of Science, South Africa*, vol. iv., p. 111 (1905).

can experience proves that the negro is worse off when crowded in cities, and that his criminality in the large centers of population is far higher than that of any other race. There is no reason to doubt that the same result would occur in South Africa.

As a matter of fact, we are already acquainted with the "location Kaffir" on the Rand, for there are already several small "locations" near the mines. The life here seems to be disastrous to the negro. He seems to lose those fine qualities which distinguish the better tribes of Kaffirs, and to pick up all the vices of the white man without gaining any of his virtues. From a view-point of health, the Kaffirs suffer, and do not thrive as well under the "location" conditions as in the free life of their kraals.

I know of one experiment of placing Kaffir families near the mines which proved a failure. At one of the outlying mines of the Rand, where there is plenty of spare land and water for agriculture, a few families of Damaraland natives were placed some years ago, during the last war between these natives and the Germans. When they went to this mine, the Damaras knew they must remain there, since they could not return to their native land. The manager of the mine informed me that the experiment was a complete failure. It seemed to be impossible to get the natives to work continuously. For generations past the Kaffir has been accustomed to work spasmodically, and the efforts of mining-men to turn him into a steady worker have so far proved unsuccessful. This failure on a small scale does not prove a great deal, but it is not beneficial to either race to live in close proximity. Many authorities on the question of the natives are of the opinion that it would not be the best policy for South Africa to bring the natives to the mines and place them in large reservations, but that they should be kept on reserves remote from the mines.

Into the complications of the present native reserves, as they are affected by the land laws, we need not enter. Suffice it to say, that the reply to those who object to the native reserve in remote parts of the country, on the ground that the Kaffirs will get such a hold of the land of the Transvaal that the white man will have no chance, is that this difficulty is exaggerated. Of the 71,000,000 acres in this colony, the natives possess about

2,000,000 acres, or approximately 3 per cent. What is wanted is a fixed policy, and to demarcate the native reserves once and for all.

It is proposed to place any future native reserves in sections of the Transvaal where the Kaffir appears to flourish, but where the white man cannot thrive, as in the northern and eastern sections of the colony. He is to be kept out of those parts especially suitable for the white man. In the demarcation of any future native reserves special efforts are to be made that the land allotted is not large enough to permit the Kaffir to live in idleness on the proceeds of women's labor, but just sufficient to enable the family to live during the absence of the man. If in years to come the Kaffirs require an extension, the land allocated to them must either be purchased or leased. The indiscriminate sale of land to the natives must not be allowed.

Since the government collects $4.85 rent per hut per year from the Kaffirs living on the reserves, the revenue of the country is aided by the reservations.

It is believed that natural economic stress will force the natives out from their reserves in increasing numbers. The case of Basutoland is one of the best examples of a native reserve in South Africa, since in the 10,293 sq. miles which make up Basutoland, there is only a handful of white traders, missionaries, and officials among a native population of 347,731. The number of Basuto boys going out to work is increasing every year, showing that economic laws are gradually pushing them into the labor market, without any interference with their native customs.

Another problem in connection with Kaffir labor is the question of supplying liquor to natives. Before the war, although it was illegal to sell or give alcoholic beverages to Kaffirs, the law was more honored in the breach than in the observance, and the week-end at the mines was a pandemonium. The surrounding veldt was strewn with drunken Kaffirs every Sunday. Serious outbreaks occurred every week between intoxicated Kaffirs belonging to rival tribes, and many lives were lost in the rows. On Mondays and Tuesdays it was no uncommon thing to find from 20 to 30 per cent. of the Kaffir labor utterly unfit for work in the mines on account of the drunken bout of the previous day. The loss to the mines due to the demoralization of the

Kaffir labor through drink amounted to thousands of dollars every year. Had the state of affairs which existed on the Rand from 1892 to 1899 continued, the Kaffir races in South Africa would have become so demoralized through drink as to be a serious danger to the peace of the country.

After the war, the British government, realizing the serious danger of liquor to Kaffirs, determined to stamp out the evil, and passed one of the most stringent laws against the supplying of liquor, not only to natives but also to colored persons and Asiatics as well. By this law any one who sells or gives alcoholic liquors to natives is, on conviction for the first offense, sentenced to prison for six months, without the option of a fine. The punishment for repeated offenses is more severe.

The enforcement of this law has had a remarkably good effect. Of course, there is some drinking done by Kaffirs, especially on Sundays, but the awful orgies of pre-war days no longer take place, and the working-percentage of the Kaffirs on Monday and Tuesday is not reduced by 30 per cent. as was formerly the case. Much of the drink of to-day is supplied to the Kaffirs at the small reservations by low Kaffir women, who are expert at evading the police. There are some alcoholic beverages which the Kaffirs make from indigenous roots and herbs, and during the week many of the women manufacture this stuff and hide it in pots, in holes dug in the earth. Unless the police know the tricks of the trade, they will visit these huts, make a careful search and then report that there is no liquor there, while at the same time within a hundred yards of the huts many gallons of the stuff is hidden. The usual way to find the liquor is to employ native detectives.

On Sundays, Kaffirs from the compounds visit these spots and buy up most of the beverage which the women have made. As a rule, they get intoxicated if possible and seldom carry the stuff back to the compounds with them. According to law it is an offense for a Kaffir to be found in possession of liquor, but if he is found intoxicated no law is broken.

These "beer drunks" on Sunday are not now the serious affairs they were in pre-war days. It is seldom now that there are any tribal fights, which formerly were of almost weekly occurrence and usually resulted in the death of several Kaffirs. From a spectacular point of view these fights were well worth

seeing, but no manager allows them if he can help it. At the first intimation that the white mounted police are coming the combatants disperse in all directions. The police have every assistance from the mine-managers, and now and then, especially about Christmas time, a squad of mounted police make a thorough search of the compounds and take away from the Kaffirs all weapons, such as spears, sticks, knives, etc., which reduces the danger of inter-tribal fights.

Under the liquor law the mining companies are allowed to supply the Kaffirs with about 2 pints per week of what is known as "Kaffir beer"—a beverage which must not contain more than 2 per cent. of alcohol. If it does, the law is broken and there is the danger of its penalties. This Kaffir beer is always made in the compounds, and every brew is tested to see that the alcoholic limit is not exceeded. It is made by adding to water 180 lb. of Kaffir corn mixed with 50 lb. of mealie meal, the whole being boiled in a pot for several hours. The mixture is allowed to cool and settle, and is strained, and yields 56 gal. of Kaffir beer ready for consumption. It requires a large quantity of this stuff to intoxicate, but the Kaffirs are never given sufficient for this purpose. They are not allowed to save up the beer, but must drink it when it is supplied to them.

Practically every compound on the Rand supplies this popular beverage to the Kaffirs, and it is also much used in the Kaffir kraals. In moderation it is beneficial, and is approved by the physicians of the Rand.

The liquor law as it now stands is by no means universally approved. A government commission now sitting in Johannesburg has had some varied and interesting evidence on the supplying of liquor to the natives. The evidence of the majority is that it would be a great mistake to tinker with the present liquor law, and make it possible for the natives to obtain liquor. The South African Native Affairs Commission of 1903–1905 strongly urged the necessity of keeping strong drink away from the natives at any cost.[2]

The upholders of the law have pointed out that constitutionally it is impossible for the Kaffir to be a moderate drinker. He is a creature of violent passions and of little control, not

[2] *Report of the South African Native Affairs Commission of* 1903–1905, p. 74.

yet out of the savage state, and if it were possible for him to obtain an unlimited supply of alcohol there is little doubt he would follow the example of the American Indian and drink himself to death, although in the case of the Kaffir the process would take much longer. The negro readily acquires a taste for liquor, and when he becomes a drinker he will do anything to get it, and if better stuff is not available he will indulge in the vilest concoctions. Kaffirs readily drink methylated spirits if nothing else is available. I have known of cases where boys have continued to drink alcoholic liquors at one sitting until they died of the effects. When intoxicated the Kaffir is a dangerous man, and in a country where Kaffirs preponderate, and in which their labor is of so much importance to the industrial development, it is necessary that they be absolutely deprived of liquor.

The opponents of the liquor law maintain that in spite of all laws the natives still buy liquor, sometimes of the poorest quality. They know the law and the penalty for having liquor found on them, and when they buy liquor they drink it all at one sitting. Were the law repealed they would become moderate drinkers and get decent liquor, instead of the present stuff which maddens them. In spite of the stringent liquor law, hundreds of cases of illicit liquor-selling come before the courts every month.

It is for these reasons that H. H. Jordan, a well-known magistrate of Johannesburg, in his evidence before the Mining Regulations Commission, maintained that the number of convictions for illicit liquor-dealing proved that the Kaffirs were securing drink, in most cases of an inferior quality. He believed it should be made legal to give mine Kaffirs a "tot" of brandy (their favorite drink) every day on coming out of the mine. The Kaffir craves for spirits of some kind, something sound and strong, but does not take to ordinary beer. Mr. Jordan was of opinion that the daily issue of brandy to the Kaffirs would be a legitimate and attractive addition to the conditions of labor on the Rand. It is maintained that if the native were allowed to buy liquor he would spend much of his money and remain on the mines indefinitely. The Kaffirs from Portuguese Africa are allowed to buy Portuguese wines freely in their own kraals and villages; they also distill for

themselves a variety of spirits from fruits and vegetables of various kinds, and are quite accustomed to these beverages. They consider it a great hardship when they come to the Rand, that they are entirely deprived of all wines and spirituous liquors.

The advocates of the "teetotal" policy for natives were much wrought up over a bill that the Boer ministry brought before the House at Pretoria last session. The object of the bill (known as the Beer Bill) was so to amend the present liquor law as to make it legal to supply spirits to Kaffirs, under government control. The government was desirous to get the bill passed, and endeavored to rush it through at break-neck speed, but such a howl of indignation went up, not only in the Transvaal but all over South Africa, and even in England, that the government very wisely but somewhat sullenly withdrew the bill. On more than one occasion, however, it has been intimated that the retreat is not final, but that the bill will be brought up again.

There will be a big fight in the future regarding the supply of liquor to natives. The mine-owners and mine-people are on the whole opposed to any change in the present law. To the argument that there are a large number of convictions for illicit liquor-dealing, they reply (1) that the law must be more rigidly enforced; (2) that if necessary a breach of the illicit liquor regulations should be followed by deportation; (3) that the painstaking, cautious South African Native Affairs Commission of 1903–1905 strongly opposed the sale of liquor to natives, and (4) that the consensus of opinion is in favor of total prohibition.

The better class of native chiefs are opposed to the supply of liquor to natives. One influential chief in Basutoland declared that he would not allow his boys to come to the Rand if it were made legal to supply them with liquor, for he well knows the danger of creating a craving for drink by this doling out of "tots" of brandy every day.

The natives themselves would welcome a change in the present laws. To the vast majority the ideal of paradise is to be permitted to drink Cape Dop (brandy) until they can drink no more. Whether the pure-blooded native and the "colored people" (who in the eyes of the law are treated like the Kaffirs so far

as liquor is concerned) will be successful in their demand that they no longer be treated as children, but be allowed the same freedom in the use of spirituous liquors as the white man, is a question to which only the future will reveal an answer.

Another important aspect of Kaffir labor is that which deals with the vital statistics. I have pointed out elsewhere in a paper, The Chinese on the Rand,[3] the heavy mortality of some Kaffir tribes that come to the mines to work. Of recent years the medical officers and managers have given much attention to the health of the Kaffirs, and most encouraging results have been obtained. Although the health of the Kaffirs may be improved still more, it must be remembered that the present Kaffir death-rate of about 34 per 1,000 per annum is quite normal for the negro race when a comparison is made with other parts of the world.

The health of the Kaffirs is well looked after before they reach the mines. Those who come from unhealthy districts, such as British Central Africa or Mozambique, are kept under medical inspection at the central compound in Johannesburg for at least 30 days before they are allowed to go to the mines to work underground.

Years ago one of the greatest scourges of the Kaffirs was small-pox[4] (introduced by the white emigrants), which spread far and near and wiped out whole tribes. So thorough is the practice of vaccination to-day, that among the Kaffir mine-laborers this disease is almost a negligible quantity. It is most liable to appear on the Rand during the cold months of June, July, and August, but epidemics are now unknown.

The Kaffir is accustomed to go "naked and unashamed," and one of the most difficult tasks for the mine-managers is to induce him to wear sufficient clothing. The Kaffir considers clothes a superfluity and thinks it a waste of money to buy garments. In the warm climate of his native kraal clothing can be dispensed with, but on the high veldt, near Johannesburg, it is a necessity. On the whole, the Kaffir mine-laborers are insufficiently clad all the year round. Many of them walk about the surface in a semi-nude state, and underground they frequently remove all clothing.

[3] P. 553, this volume.

[4] Theal's *History of South Africa.*

This lack of clothing is the cause of much of the illness among Kaffirs, especially those from malarial districts. In the winter months Kaffirs often come out of the mine in a heated state, with very little on them, and find a high cold wind blowing. Before long they are chilled through and through, and, of course, are soon in the hospital with severe colds or pneumonia.

This question of insufficient clothing has such a direct effect on the death-rate among the Kaffirs that I believe the government should pass a law compelling the Kaffir to obey the rules of health and clothe himself properly. A regulation of this kind would do more than any other one measure to reduce sickness and death among the Kaffir mine-laborers.

During recent years there has been some improvement in the clothing of the Kaffirs. Boys from the East Coast are supplied with clothing before they come to the Rand. But the whole trouble lies with the Kaffir himself. For generations he has gone naked and barefooted, and he does not care to spend his money for clothing or boots. It is of course impossible to expect the companies to clothe him, but if a compulsory clothing law were passed the mine-officials would give every assistance in its enforcement.

All the world over the negro race shows a high death-rate, so that the racial tendency must not be lost sight of in studying the vital statistics of the Kaffirs. At the mines the death-rate of Kaffirs varies greatly from month to month, as is shown in Fig. 1,[5] but the improvement has been steady on the Rand. Although the death-rate among natives from the hot, malarial districts is still high, there has been a decrease during the past two years. Care is taken to bring these natives here during the warm months, so that they can become acclimated before winter begins, and recently the British government has decided, on humanitarian grounds, to prohibit the recruiting of Kaffirs for the mines of the Rand from British Central Africa, for in spite of all precautions the natives from these parts show a very high death-rate when removed to the Rand.

In a paper[6] by L. G. Irvine and D. Maccaulay it is pointed

[5] *Report of Native Affairs Department* (*Transvaal Administration Reports*), 1905–1906, p. 127.

[6] The Life History of the Native Labourer, *British Association for the Advancement of Science*, vol. iii., p. 342 (1905).

out that the mortality on the mines can be explained under two headings:

1. *Territorial or Climatic, and Racial Factor.*—The statistics, given by Drs. Irvine and Macaulay, Table II., show the relative rates of mortality of the natives from different parts of Africa during the year ending June, 1905.

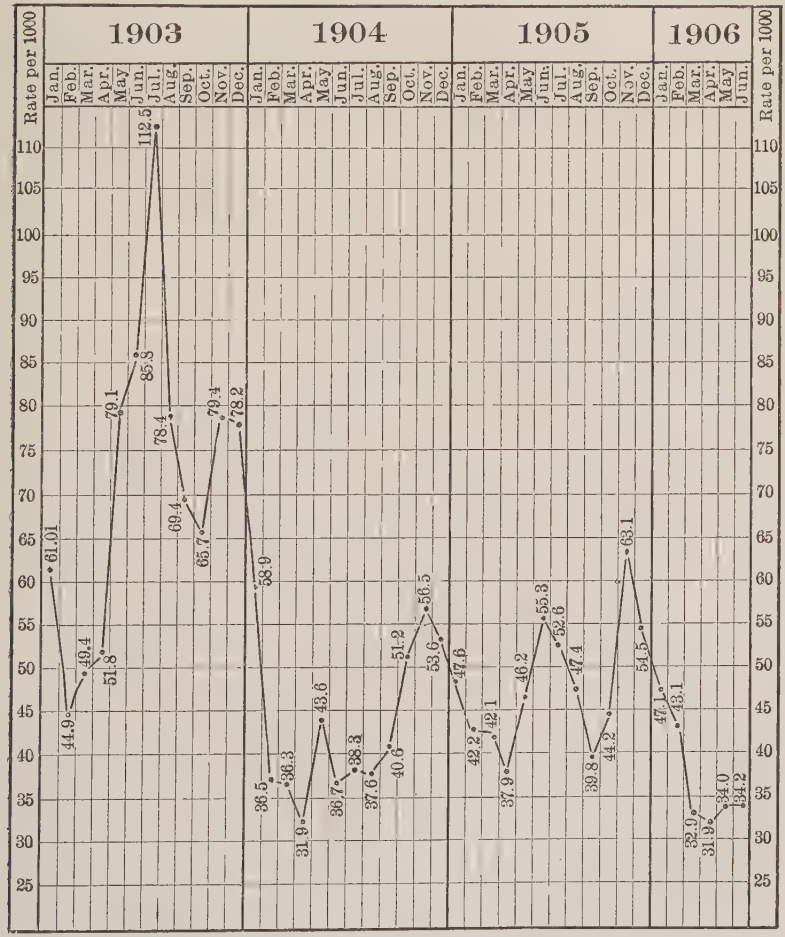

FIG. 1.—MORTALITY-CHART, SHOWING THE DEATH-RATE PER 1,000 PER ANNUM FOR EACH MONTH AMONG NATIVES EMPLOYED ON MINES AND INDUSTRIAL WORKS IN PROCLAIMED LABOR DISTRICTS.

The statistics show that the natives from those parts of South Africa where the climate resembles that of the Rand have a low mortality, while the Kaffirs from the sub-tropical and malarial districts have a high death-rate. It is fair to state

that the mortality of the Kaffirs from malarial districts improves after a sojourn of a few months. A large number of the deaths occur during the first month after their arrival. But in spite of all precautions, the death-rate of natives sent to the Rand from British Central Africa continues so high that the British government has forbidden the recruiting of boys for the Rand from this part of Africa.

TABLE II.—*Mortality Statistics of Natives for Year Ending June*, 1905.

	Death-Rate Per 1,000.	Complement.
Natal and Zululand show a death-rate of	12.8	2,579
Cape Colony,	17.3	10,104
Basutoland,	28.2	2,798
Transvaal,	38.6	7,400
Southern East Coast Provinces,	39.9	49,396
British Bechuanaland,	41.2	1,043
British Central Africa,	118.3	1,463
Rhodesia,	118.5	2,395
Northern East Coast Provinces,	.....	.....
Mozambique,	138.2	1,630
Quilimane,	163.9	1,458

2. *The Occupation Factor.*—As could be expected, the health of the Kaffir working on the surface is far better than that of the underground worker. On the Rand the relative proportion of Kaffir underground workers to surface laborers is about 2 to 1, but an investigation of the mortality statistics shows that the mortality is not as 2 to 1, but as 5.5 to 1. Of the total deaths 85 per cent. occur among the underground laborers.

There are a number of reasons for the higher mortality among underground workers. The average mine-temperature is from 70° F. to 75° F. for the year. During the cold months, when the Kaffirs come up from the mine, in most cases illy clad, they get chilled through and sicken. In the opinion of Drs. Irvine and McCaulay the compulsory use of change-houses for Kaffirs and the insistence on changing their clothing when they come from the mine, and wearing sufficient garments, would do a great deal to reduce their high death-rate. Then there are the risks to health of underground workers due to the vitiation of mine-air by noxious gases from explosions. Some of the cases of acute pneumonia are undoubtedly due to the poisonous nitrous oxide fumes. Insufficient ventilation in sec-

tions of the mines, before connections are established, accounts for the illness of some of the Kaffirs.

On the whole, the mines are dry and dusty. Kaffirs who work on machine-drills swallow a great deal of dust every day, and in time the dust produces silicosis. The regulations regarding the use of sprays on machines, and dampening-down of the ore with water before shoveling, are only partly carried out, so that the injury to the natives' health through excessive dust must be taken into consideration. Neither white men nor Kaffirs use as they should the appliances provided.

The Kaffirs are not as healthy as the white race. One authority,[7] G. Liengme, declares that fully 30 per cent. of the native children die before they reach the age of two years. It is sometimes difficult to treat the Kaffirs on account of their ignorance. They attribute every ill to some supernatural agent, either witchcraft or deviltry, and much prefer the witch doctor of their kraals to the medical practitioner. They are excellent subjects for suggestion, and Dr. Liengme states that he has often hypnotized patients to perform surgical operations. Of the individual diseases, by far the greater number of deaths are due to pneumonia. The Kaffir is susceptible to all pulmonary complaints. The mine-managers pay great attention to the health of the Kaffir while he is at work, and each mine has a well-built hospital in charge of a European doctor, who, as a rule, is also the physician of the whites employed at the mine. Besides housing and rations, all natives get medical and hospital attendance free of charge. They are given a special diet of milk, bread, coffee, etc., when in the hospital. The physician can order stimulants for them if he considers it necessary.

The accident-death rate among Kaffirs is very high, in spite of the most elaborate mining-laws and regulations in the world. They are both ignorant and reckless, and do not take care of themselves while working underground. A Kaffir is recompensed for total, part, or temporary disablement, and when killed at the mine his heirs receive $50.

One of the most important points in the successful working of Kaffirs is to have the right sort of compound-managers.

[7] Diseases Amongst Natives, *South African Association for the Advancement of Science*, p. 368 (1905).

Some men have a remarkable power over the Kaffirs, and many natives prefer one mine to another, because at one there is a white boss for whom they have more respect. The natives are accustomed to strict, just treatment. The successful compound-manager never pampers them, nor is he familiar with them. He lends an ear to their complaints, and redresses any wrongs to the best of his ability.

The Kaffirs soon learn the discipline of the compound. The compound-manager has an assistant and a number of Kaffir policemen to help him in his arduous duties. The police boys are well paid and are selected from the tribes they are to "boss up." As a rule, it is not a good policy to place a Zulu policeman in charge of a crowd of Xosas. The natives prefer a policeman of their own tribe to be over them. A general view of a Kaffir compound is given in Fig. 2.

Compared with the Chinese, the Kaffir spends very little of his wages. Some tribes save more than others. The natives from Portuguese East Africa, for instance, who sign on for a year, spend a fair amount of their money, especially on liquor, while the natives from Cape Colony, who, as a rule, work for only six months, hoard their savings.

The Kaffirs are signed on to work 10 hr. per day, day- and night-shift as required. The shift starts at 7 o'clock and ends at 5 o'clock, with an hour for dinner. On a few mines the 8-hr. shift is in force. The Kaffirs are awakened in the mornings about 5 o'clock. At some mines they are given hot coffee before going out to work, especially during the cold months, but at others the Kaffir has nothing for his breakfast but mealie porridge left over from the previous night.

As a matter of fact, the native is trained to one meal a day, and, like a camel, he can take in enough at one sitting to last him for a long time. Few men individually can eat as much at one meal as a Kaffir. For several hours after he comes up from work the Kaffir literally stuffs. Some take a bit of mealie porridge into the mine to eat at mid-day, but the majority go through the shift without food.

The Kaffir is a proof of the value of the American Indian corn (or mealie) as a food. From childhood mealie meal is his principal food, and at the mines it constitutes the chief article of his diet. In the Southern States of America the ex-

FIG. 2.—GENERAL VIEW OF KAFFIR COMPOUND.

FIG. 3.—A MACHINE-DRILL AT WORK UNDERGROUND.

FIG. 4.—SUNDAY MORNING IN A COMPOUND.

FIG. 5.—SUNDAY MORNING IN A COMPOUND. KAFFIR PIANOS IN FOREGROUND.

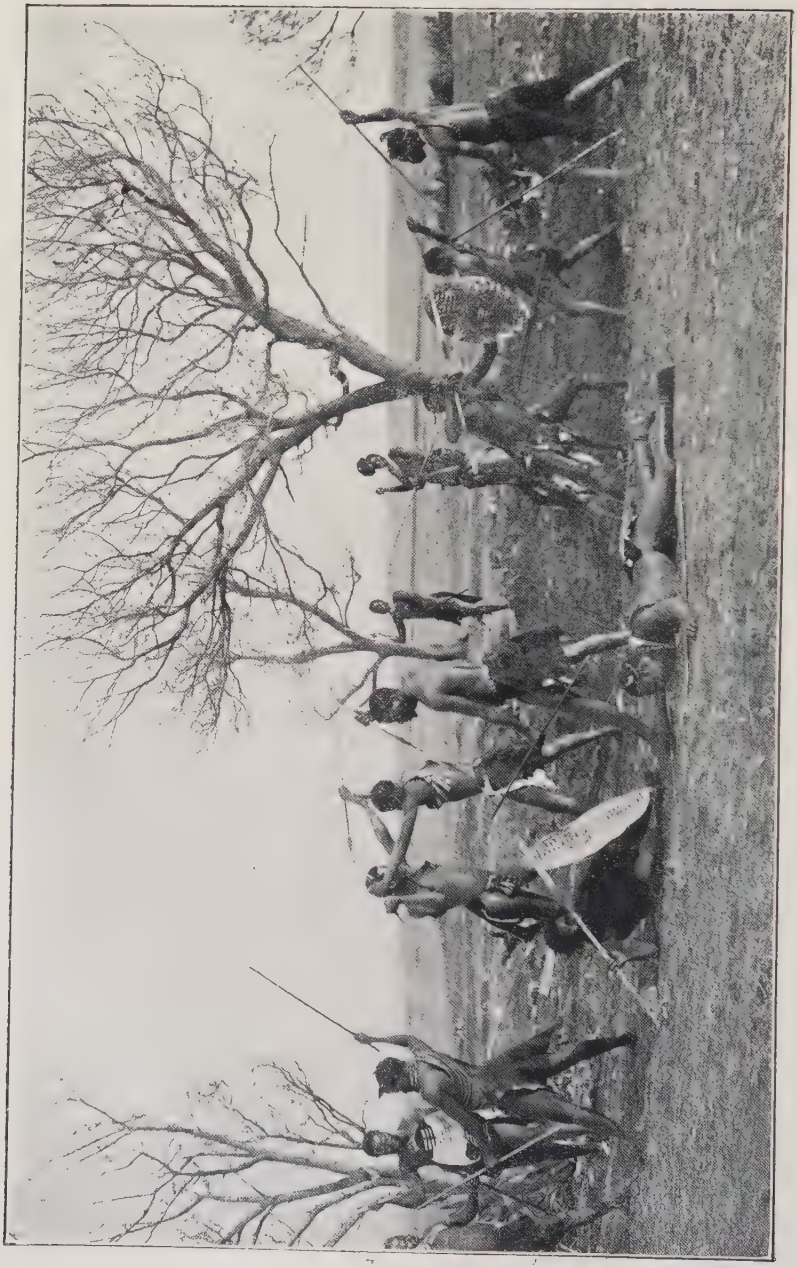

FIG. 6.—KAFFIRS PRACTICING AT WAR.

perience with corn meal is the same. There, however, the white population consumes tons of corn meal every year, whereas on the Rand, where there is an abundance of corn, white people have had an absurd prejudice against mealie meal because it is "Kaffir food."

Reference has already been made to the use of Kaffir beer. This beverage is not only drink but food as well, and is an important item in feeding the natives. The boys receive from 1 to 2 lb. of meat per week, and, as a rule, prefer to have it in bulk on Sundays, when they cook it themselves. Vegetables are also given to them once or twice a week. At some mines a vegetable stew is given every day, and for variety beans are also given.

In the early days of the Rand little attention was given to the diet of the Kaffir, but men realize now that to get good work out of a laborer he must be fed well. Since the advent of Chinese there has been an improvement in the feeding of the natives, and to-day they can find no complaint about the food given in the compounds, as regards either quantity or quality. It is plain but wholesome.

The better feeding has practically banished a disease that was once common—namely, scurvy. Some years ago a large percentage of the mealies was imported, but now nearly all of the mealie meal used on the mines is South African grown. The imported mealies seem to have produced scurvy in a number of cases.

When the "green" Kaffir first arrives at the mines he is of little use, and is as ignorant of mining as a monkey. In starting him to work he is placed, whenever possible, with experienced Kaffirs of his own tribe. Some differ from others as to the time allowed the Kaffir to learn his work. After the time of probation has expired he must do a certain amount of work, or receive no pay for the day. For many years 36 in. was taken as a day's work for a "hammer" Kaffir, irrespective of the hardness of the rock on the different mines. The consequence is that at mines where the rock is not over hard the majority of hammer Kaffirs finished the day's work by midday. The Chinese experience has shown that the Kaffirs were doing about half a day's work. Every effort is now being made to increase the depth drilled per shift by the Kaffirs, and,

as I stated before, the Kaffir is being induced to drill more by being placed on the contract and bonus system.

At the French Rand mine a Kaffir is allowed two weeks in which to learn how to drill. For this period he is paid the rate at which he was recruited, from $0.40 to $0.50 per shift, as the case might be. After a fortnight's training he is put on contract at 1 cent per in., and unless he drills at least 24 in. he receives no pay. Some Kaffirs are now drilling two holes per shift, for which they receive $20 and more per month.

Until he learns the work the Kaffir is of little use, and is only an expense to the companies. The short period of service is most unfortunate from the point of view of the mines, for an inexperienced Kaffir on hammer-work often takes nearly two months to become really efficient, so that the company gets only four months' good work from the mine-manager's point of view. One of the most discouraging and heart-breaking features of Kaffir labor is the fact that just as he gets thoroughly accustomed to the work and becomes useful he leaves you. His place is, as a rule, taken by an ignorant Kaffir, and the process of "breaking in" must be repeated. The necessity of longer contracts with the Kaffirs is apparent.

On machine-drills a Kaffir proves a good workman after he has learned the work. His white boss, who is on contract, frequently gives him a small bonus at the end of the month, if he works well, and in this way the "machine boy" is induced to "get a move on." A view of a machine-drill at work underground is given in Fig. 3.

In shoveling and tramming the Kaffir is at his best as a mine-laborer, and does good work either on contract or day's pay. In timbering, on skips, etc., he is also satisfactory.

A most important point in the successful management of Kaffirs underground is the character of the white boss who is in charge of them. It is more important to have competent miners on the Rand than in other parts of the world, for if the man in charge of the Kaffirs is incompetent it means that the labor of all the Kaffirs will be inefficient. There is a knack in "running" Kaffirs. With one white man a gang of Kaffirs work well, and there is never any complaint, while with another man there is continual friction and complaint. On the whole, the Kaffir is the most docile laborer in the world, and a white

boss should have no difficulty in controlling him. The majority of the natives instinctively recognize the authority of the white man.

The number of Kaffirs that a white man is able to "boss up" varies. If he has a crowd of inexperienced hammer boys in a difficult and dangerous place he will be kept busy with 18 or 20. In favorable stopes and with well-trained Kaffirs, a white man may "boss up" from 45 to 55 natives. A white man who works two rock-drills has four or five Kaffirs to help him; and with three drills he has six or seven. Few white men work in the mines without a Kaffir of some sort nearby to help them.

The ratio of the total number of white men employed at the gold-mines to the number of Kaffirs is 1 white to from 8 to 11 Kaffirs. The percentage of Kaffirs working varies from 89 to 95 per cent. Sickness accounts for the largest percentage of absentees. At the French Rand mine 74 per cent. is at work underground and 26 per cent. on the surface. At the coal-mines, the ratio of the white men employed to Kaffirs is very small. One white man frequently looks after from 80 to 90 Kaffirs in the mine, which no doubt explains the low costs of operating Transvaal coal-mines.

The mode of communication between the black man and the white is a *patois* known as "kitchen Kaffir." It is a lingo made up of many parts, English, Dutch, etc., but the majority of the words are of Zulu origin. It is understood by all the Kaffirs working on the mines. As there is no grammar the dialect is readily picked up by the white man, and even the stupid natives from Central Africa learn it after they have been a short time at the Rand mines.

Recently there has been a distinct improvement on the Rand in the recovering of native deserters from the mines. These deserters are punished by imprisonment or a fine, and are then sent back to the mines. It is felt, however, that an improvement is required in the recovery of deserters. When the finger-impression department is put on a sounder basis it is believed that the majority of the Kaffirs who run away from their work will be found and brought back.

As a surface laborer the Kaffir is more satisfactory, for his health is far better when he works above ground. He is put

in the blacksmith-, fitting-, and carpenter-shops as a helper to the white man. In the mill and cyanide works he does very well, but sometimes gives trouble on account of his thieving propensities. Much of the gold-stealing that goes on is done by Kaffirs, and for this reason many managers believe in an "all white" policy in the departments where honesty is absolutely essential.

The vast majority of the Kaffirs have Sunday as a holiday. The law allows only necessary work to be done on this day. Missionaries are allowed to teach the Kaffirs on Sundays, but the majority of the boys prefer their amusements of war-dancing, piano-playing, etc. On the whole, the Kaffirs are content with the treatment they receive at the mines, and to a visitor they appear a very happy lot. Views of Kaffirs in a compound on a Sunday morning are given in Figs. 4 and 5, and Fig. 6 shows a group practicing at war.

Kaffir labor is more plentiful to-day than ever before in the history of the mines of the Transvaal. At the present time there are about 169,000 Kaffirs employed at mines and works; there are also 26,504 indentured Chinese at work. All the mines are fully supplied with natives, and thousands more could be obtained if they were required. The companies have wisely taken advantage of this plethora of labor to insist on longer service in their contracts with the natives. Instead of the usual six months' agreement, the Kaffirs are now signing for nine months and for a year.

The cause of this abundance of labor in the gold-mines is well understood. The diamond-mining industry is in a very depressed state, so that there is a restricted output of diamonds. In consequence, thousands of Kaffirs have been thrown out of employment. The railroads are not being extended, and the different governments are keeping down expenditures as much as possible. The only field left for the natives is that of the gold-mines. Whenever there is a general industrial awakening throughout South Africa, it seems probable that there will again be a shortage of labor for the mines.

Investigation on Jigging.*

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(Chattanooga Meeting, October, 1908.)

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I. Introduction.

The jig, in one form or another, continues to hold a leading place among the machines designed to separate two or more

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minerals of different specific gravities. It is simple in construction, easily operated, capable of treating large quantities in a short time, and highly efficient under various conditions.

The question, whether the material to be jigged has first been sized, determines the two principal methods of jigging. Jigging preceded by close sizing, generally known as the Continental or German system, involves a more or less elaborate series of screens or trommels, with attendant cost for installation, operation, and repairs. Jigging without sizing, known as the English system, is, according to Munroe,[1] "a development of the hand-jigging formerly employed in Cornwall . . . and introduced by English miners to this country." In its simplest form, the method consists in jigging an ore-mixture previously crushed to some maximum size (although, in some cases, even this preliminary is omitted) on a relatively coarse sieve, and then jigging again on a finer sieve, the material passing through the first sieve and bedding. While many modifications have been necessary to adapt it for use in mills of large capacity, where hand-work was necessarily replaced by machines, the principle remains the same; the fact that the English system has been successfully employed, both in this country and abroad, is well known; and arguments have been made for its efficiency and applicability in a wider sphere than it has occupied hitherto.

II. Review of Previous Investigations.

The fact that treating a mixture of minerals under jigging conditions increased the amount of mineral saved; or, as Professor Richards aptly terms it, "the extra jig-catch," has long been known. To account for this fact a number of theories have been proposed. The work of Rittinger[2] in this field has, for many years, been a classic in the literature of ore-dressing. For the purposes of my present paper, however, the work of two American investigators, Prof. H. S. Munroe and Prof. R. H. Richards, is chiefly concerned.

Professor Munroe has given the results of an elaborate series of experiments,[3] and his deductions, based largely on theoretical grounds, of this work. After reviewing briefly the two systems of jigging, followed by a discussion of Rittinger's formulas and

[1] *Trans.*, xvii., 637 (1888–9).

[2] *Lehrbuch der Aufbereitungskunde*, pp. 165, 270 (1867).

[3] *Trans.*, xvii., 657 (1888–9).

the derivation of them, and after a careful study of the behavior of grains (usually shot) in a tube *en masse*, acted upon by a rising current of water, he is led to conclude that the interstitial currents play a very important rôle, and are responsible for the high ratios of concentration obtainable in the English system of jigging. Since his conclusions bear directly upon the present investigation, they are given in full, as follows:

"1. Bodies falling through water in a tube do not attain as high a velocity as in falling through the same medium in large vessels.

"2. The falling velocity is but little affected when the diameter of the body is less than one-tenth that of the tube.

"3. The falling velocity is the more retarded as the diameter of the body approximates that of the tube.

"4. A sphere four-tenths the size of the tube will develop the greatest falling velocity, and will require a current of maximum velocity to support or raise it.

"5. Grains falling *en masse* are really moving in confined channels, and follow the law of the movement of bodies in tubes. The falling velocity, and the velocity of the current necessary to support or raise a mass of grains, increase and diminish with the distance apart of the grains.

"6. The diameter of the channel in which the single grain moves equals the cube root of the volume of the grain with its proportion of the interstitial space. . . .

"7. In a mass of grains of different sizes, the large grains move relatively in smaller channels than the small grains. The ratio of the diameters of equal-falling grains of quartz and galena, under such conditions, is 31 to 1, instead of 4 to 1, which latter ratio holds good for free-falling grains only.

"8. The formulæ for grains moving in tubes, when applied as above to grains moving *en masse*, enable us to compute the velocity of jig-currents and thus determine the proper length and number of strokes of the jig-piston. The old formulæ gave results many times too large.

"9. *The present investigation demonstrates that close sizing is not necessary for the separation of different minerals by jigging, unless the difference in specific gravity is small.* . . .

"10. Downward currents are apparently necessary to success in jigging through a bed. This requires confirmation by experiments on a larger scale.

"11. Very fine material, less than $\frac{1}{10}$ millimeter in diameter, can be treated successfully on jigs, *if treated with coarse stuff*, the concentration taking place in the small interstitial channels between the grains forming the mineral bed. *For the treatment of fine stuff on jigs, close sizing is a positive disadvantage.* Jigs work well on mixed stuff, and very badly on fine stuff alone. Stuff less than four-tenths the size of the smallest interstitial channels cannot be treated successfully in this way.

"12. The size of the mesh of the jig-sieve has a very important influence, and must be proportioned to the work to be done.

"13. The English method of jigging without sizing, except possibly so far as is necessary to remove the very finest slimes, has many advantages, and should be more generally adopted."

Professor Richards,[4] in a very careful and elaborate investiga-

[4] *Trans.*, xxiv., 409 (1894).

tion on the question of jigging relatively small sizes, treats it under four heads: (1) the law of equal-settling particles; (2) the law of interstitial currents; (3) the law of acceleration; and (4) the law of suction. These four laws are supposed to govern all jigging operations. Practically, Professor Richards's full conclusions are:[5]

"The two chief reactions of jigging are pulsion and suction.

"The effect of pulsion depends upon the laws of equal-settling particles, interstitial currents, and, possibly, also of acceleration. The chief function of pulsion is to save the larger grains of the heavier mineral, or the grains which settle faster and farther than the waste.

"The effect of suction depends upon the interstitial factor of the minerals to be separated. . . . If this factor is greater than 3.70, suction will be efficient and rapid. If the factor is less than 3.70, suction will be much hampered and hindered. The use of a long stroke will help to overcome this difficulty. The chief function of suction is to save the particles that are too small to be saved by the laws of equal-settling particles, and of interstitial currents, acting through the pulsion of the jig.

"For jigging mixed sizes, pulsion with full suction should be used.

"For jigging closely-sized products, pulsion with a minimum of suction should be used."

He concludes by saying, in effect:

In jigging minerals having an interstitial factor greater than 3.7, sizing is simply a matter of convenience, although the fine sizes should be removed in some suitable manner. But if the factor is less than 3.7, then the jigging of mixed sizes cannot give a perfect separation, and if this is desired, then close sizing must be adopted, and the closer the sizing the more perfect the jigging. As an expedient, however, there are often cases where a satisfactory separation may be attained without sizing.

The differences in the conclusions of the two investigators above quoted have been chiefly influential in suggesting this present investigation, which was begun in the fall of 1906, and the results of the work done in the Mining Laboratory of the Columbia School of Mines have been embodied in a paper submitted to the Faculty of Pure Science in Columbia University. Since most of the work done then was preliminary to that recently undertaken, I include herewith a *résumé* of my former results and conclusions.

III. Résumé of the Results of Previous Preliminary Work.

In the following investigation an effort was made to determine, among other things: (1) the conditions and laws of

[5] *Trans.*, xxiv., 485 (1894).

hydraulic classification; (2) the conditions and limitations of jigging in the pulsion-jig; (3) the effect of varying the length and number of strokes per minute in the Vezin laboratory-jigs; (4) experiments with a large five-compartment Harz jig to determine the limits and perfection of separation effected in an ore containing galena and sphalerite with a quartzose gangue.

Considered briefly, the results of these tests, in the above-named order, are:

1. *Hydraulic Classification.*

A number of tests were made with quartz paired with galena, antimony, arsenopyrite, magnetite, sphalerite, etc., in different proportions, and with a velocity varied between wide limits, in order to determine whether a fixed ratio existed as to the diameters of the grains of the two minerals. All tests under this head were made in a Munroe hydraulic laboratory-classifier. Without going into details of the methods, etc., the results indicated that whether or not a more perfect separation was effected in the classifier-tube itself, the manner of drawing off the classified products always resulted in giving a large proportion of mixed products, and after a number of calculations upon different drawings, similar to the manner detailed under the pulsion-jig tests, and described by Professor Richards,[6] proved to my satisfaction that no such ratio existed with classified products under the conditions the above type of classifier was operated and the products removed.

2. *Pulsion-Jig Tests.*

The largest size of Munroe hydraulic classifier was first fitted up in such way that a column of ore 5 to 6 in. long was supported upon a bedding of large grains, and then treated with a pulsating current of water. The tube in which the jigging took place had a diameter of about 1.75 in., and the pulsion was effected by compressing a rubber tube connecting the bottom of the ore-column with a pressure-head of water. The compression of the tube was effected both by mechanical means and by hand, and apparently it made little difference which method was used. The bedding-grains served only to support the ore-column and confine it within the tube; and in

[6] *Op. cit.*, p. 450, *et seq.*

drawing off the products this was always first to be removed. The results of jigging under these conditions and the removal of the jigged product—namely, by allowing the jigged material to subside gradually into a rubber tube connected with the receptacle which supported the bedding, if it may be called such, and which was really the hutch of the jig, were that after drying, screening, weighing, and analyzing the different screen-products from a number of drawings, and finally calculating the ratios between the diameter of the grain of quartz and that of the other mineral paired with it, no such ratio as that given by Richards could be obtained under such conditions, but the tests were in all respects duplicates of the first series run with the classifier operated under the conditions of hydraulic classification.

It was found, however, that if the jigged products were not removed from the jigging-tube as above described, but, instead, a screen attached to the lower end of the jigging-tube, and the mixture of minerals jigged on this screen, and then instead of drawing off the products through the rubber tube at the bottom the entire apparatus was dismantled, and the jigged products removed from the tube by inserting a piston and forcing the ore-column from the bottom of the tube, cutting sections at equal intervals, that approximate concordant results were obtained. These sections, which were cut off at equal intervals, and usually eight or nine in number, were dried, sized on a nest of sieves, weighed, and analyzed. Ratios of diameters were then calculated for some four or five drawings, in which the mixed grains occurred, according to the method described by Richards,[7] which was as follows: The average diameter of the quartz-grains was obtained by multiplying all the quartz-weights in a particular drawing by their diameters, and dividing the sum of the products by the sum of their weights; and similarly for the other mineral paired with it. The average diameter of the quartz-grain thus determined is divided by the average diameter of the grains of the other mineral, and the quotient is the desired ratio. Table I. gives the ratios that were obtained with the pulsion-jig, the material in nearly all cases being sized between 0.15 and 2 mm. For purposes of comparison I have included the ratios obtained by Professor Richards[8] with a

[7] *Trans.*, xxiv., 450 (1894).

[8] *Trans.*, xviv., 463 (1894).

pointed tube, the results of which he considers to hold true for the pulsion-jig as well.

TABLE I.—*Equal-Settling Ratios of Minerals in Pulsion-Jigs.*

Name and Specific Gravity.		Ratio for Pulsion-Jig.	Richards's Ratio.
Quartz, 2.58 and	Galena, 7.14.....	5.80	5.842
	Antimony, 6.66.....	5.20	4.896
	Arsenopyrite, 5.71.....	4.42	3.737
	Magnetite, 4.76.....	3.65	not given.
	Sphalerite, 3.70.....	2.61	2.127

In the tests of Table I., 50 per cent. by volume of each mineral was used. It seems evident, therefore, that under the conditions that exist under the influence of pulsion alone, the free-settling ratios obtained with Rittinger's formula[9] are increased, but by no great amount.

3. *Vezin Laboratory-Jig Tests.*

Without going into the details of construction of this very useful little laboratory-apparatus, suffice it to say that the piston is driven by a variable-speed shaft, with a disk and friction-wheel, and the number of strokes may be varied from 100 to 300 per min., and, with a double eccentric, the length of stroke from 0 to 1.25 in. (31.7 mm.). The box carrying the sieve is attached to the body of the jig by means of clamps, so that, together with the ore and bedding resting on the sieve, it may easily be removed and the contents examined, or another box with its attached sieve substituted. In all tests with the Vezin jig a sieve of 8-mesh (2.2 mm. square hole) was used. The bedding was in most cases sized between the limits of 2.5 and 3.3 mm., and maintained at a thickness of 0.75 in. (19 mm.). The jig was driven from a counter-shaft by an electric motor, so that a uniform speed was secured. The feed in all cases was sized between the limits of 0.10 and 1.9 mm., and the various mixtures were made up by volume to contain 3 of quartz and 1 of the heavier mineral. From 1.6 to 2.0 kg. represented the amount generally employed in each test. After this quantity had been run over the jig it was stopped, the sieve-box removed, the contents placed in a large pan and dried, the hutch-work

[9] *Trans.*, xvii., 639 (1888–9); *ibid.*, xxiv., 411 (1894).

drawn off, the water decanted and treated in the same way, and finally the tailings were freed as far as possible from water and dried. The three products were then sized separately on a nest of sieves, each size weighed and analyzed, the material being subsequently used again for another test.

It is evident that in so simple a machine as the Vezin jig there are a number of factors that may be made either constant or variable. Thus the length of stroke and number per min. are easily varied, or may be kept constant; the size of the grains constituting the bed, and its thickness, may be varied within limits, although this is likely to vary with other factors, especially the piston-speed; then the quantities of water used on the piston side, with the feed and the amount discharged from the hutch, as well as the rate of feed, may also be varied. In these tests the length and number of strokes were the principal variables, and also the amount of suction, of which there are a number of degrees, limited as follows:

(A) Full suction. In which the hutch-spigot is fully open, and the water thus discharged is supplied entirely by increasing the amount added with the feed, and, if possible, cutting down the amount supplied to the piston side.

(B) Part suction. In which the hutch-spigot is not fully open, and does not discharge a quantity equal to the extra amount added with the feed.

(C) Balanced suction. In balanced suction the hutch-spigot is closed and the feed-water and piston-water are equal; or the hutch-spigot is partly or fully open, and the amount thus discharged is supplied entirely from the piston side.

The results obtained indicate the following conclusions:

A. *Suction.*—With full suction, (A), the bed was not mobile, and after a few minutes' feeding the jig was very badly choked and little or nothing passed into the hutch. After trying a few tests with the same bad results, full suction was considered impracticable. In the case of part suction, (B), the mobility of the bed was decreased in proportion to the amount of suction, and with it a decrease in the amount of coarse mineral passing into the hutch, but with a corresponding increase in the amount of fine material without a noticeable enrichment. The best results were obtained with balanced suction, having the spigot completely closed, although the results with the

spigot partly or fully open did not differ materially from those of full suction (A).

B. *Feed-Water and Rate of Feed.*—These factors were kept as nearly constant as possible, and the effect of varying them was not considered.

C. *Filter-Bed.*—The thickness and the size of the filter-bed, also, were made a constant. It was found, however, that the shape of the grains of the bedding does influence the ease and rapidity with which the mineral passes into the hutch. Thus with antimony and arsenopyrite, both of which break into long, pencil-shaped grains, the sieve became quickly blinded, which interfered with the free passage of grains below, and required a long, heavy stroke to dislodge them.

D. *Length and Number of Strokes.*—The results of the tests seemed to show that the character of the separation is not directly dependent upon absolute piston-speed, but that the quick, short stroke was more efficient, and resulted in a cleaner hutch-product, and relatively more of it, than a longer stroke of less frequency, but of the same piston-speed.

E. *Concentration.*—If the diameters of the grains of the heavy mineral jigged, and of the bedding-grains (and therefore the diameter of the sieve-hole), do not differ by any large amount, a clean separation can easily be made. With an increase in these ratios, perfect separation is impossible. Stated in other words, with bedding of a definite size, and hence a fixed sieve-aperture, the finer the grain the more difficult is its separation on the jig.

F. *Specific Gravity.*—Within rather wide limits, the difference in the specific gravity of the heavier mineral paired with quartz did not influence greatly the ease with which it could be separated, or a good concentration attained.

4. *Experiments with a Five-Sieve Harz Jig.*

Two runs were made as nearly as possible under practical conditions to determine to what extent the conclusions derived from the Vezin-jig tests were applicable to an ordinary jig. The ore used for the work contained 6 per cent. of mineral—about half sphalerite and the balance galena, with a quartzose gangue. The jig was bedded with material sized between 5.2 and 6.6 mm. The first compartment was bedded with a clean galena, the

second with sphalerite, and the third, fourth, and fifth with mixtures of sphalerite and quartz. The thickness of the bedding averaged from 20 to 30 mm. at the beginning of the run. All beds naturally tended to increase in thickness, since no products were skimmed off during the run.

The jig differed in no respect from the common type of Harz jig. Each sieve-compartment was 16 by 20 in. (406 by 512 mm.) in section, with pistons of equal area. The lengths of strokes could be adjusted between limits of 0 to 50 mm., and within a considerable range in the number per min.—in the experiments, from 175 to 180. The actual piston-speeds used ranged about as follows: first compartment, 75 mm.; second, 66 to 70 mm.; third, 57 to 67 mm.; fourth, 45 to 58 mm.; and fifth, 45 to 50 mm. per sec. Only the hutch-products and tailings were examined.

The ore, sized between 0 and 4.8 mm., round hole, was delivered to the jig through a centrifugal pump. All products traveled in closed circuits, and were finally returned to the centrifugal elevator or pump to be passed again over the jig. The spigots constantly discharged their products, and from these discharges time-samples were cut out. The run occupied exactly an hour, so that after weighing each of the products—in this case six—with the tailings, data were at hand for calculating the capacities; and after screening, weighing, and analyzing, a complete record of the run was made. The results of these tests showed that the differences in length of stroke, or number of strokes per min., were not sufficient to produce a marked difference in the character of the concentrate; that most of the galena was saved in the first hutch and most of the sphalerite in the second; that the third, fourth, and fifth hutches carried very little galena, but more sphalerite. It was found that the first hutch-product contained 57 per cent. of galena, and of this nearly 70 per cent. was larger than 1 mm. in diameter; and that sizes finer than this contained more quartz and less galena. The results seemed to indicate the necessity of first removing stuff less than 0.4 mm. in diameter in order to increase the richness of the product. The first hutch-product contained no coarse sphalerite, and only when the material was as small as 0.2 mm. was any considerable amount present. This seems to indicate that an almost perfect

separation of these two minerals (galena and sphalerite) from each other and quartz, under the conditions with which the jig was operated, was possible if the feed had been sized between the limits of 0.4 and 4.8 mm. The second hutch, which carried most of the sphalerite, shows that the coarse sizes pass through the sieve of the jig less readily than galena. Not until the material was reduced to 1.5 mm. was any marked percentage noticeable in the product. From 0.4 to 1.5 mm. most of the saving was made. Evidently, the cause for so little very fine stuff in the second hutch-product was owing to the fact that most of it was caught in the first. Under the conditions obtaining in the second compartment, a very satisfactory separation could be made on all sizes below 1.5 mm. The results from the third compartment were like those of the second. The fourth and fifth hutches indicated a further saving of sphalerite, but between somewhat different size-limits than in the second and third; the limits in the last two compartments varied between 0.7 and 2.5 mm., with very little fine stuff. This result indicates that in the first compartments more fine material is present, making a denser and more impervious bed, and that the large grains cannot so easily pass through it; and that in the last compartment the bed is more open and porous, and hence larger grains can more readily pass into the hutch. An examination of the tailings indicated that the loss in the fine material was very small, but by far the largest loss was in the four coarsest sizes, which were mixed grains or middlings, and to reduce this loss further crushing must be done. The results indicate that in order to separate sphalerite and quartz, a jig of at least three compartments should be used; since smaller differences in the specific gravity of these minerals require a longer time to effect the separation. In the case of a heavy mineral, such as galena, one or two compartments will effect a perfect separation.

IV. Experiments with the Jarvis Laboratory-Jig.

In order to investigate particularly the effect of pulsion and suction upon jigging, and upon accelerated and retarded strokes, I designed a special jig, with which I conducted a series of experiments and obtained the following results:

1. *Construction.*

Figs. 1, 2, and 3 are detailed drawings of the Jarvis laboratory-jig, with the exception of the variable-speed shaft, which is of the ordinary disk-and-friction-wheel pattern. Figs. 4 and 5 show the designs of the cams used. The screen-area in this jig is 8 by 12 in. (203.2 by 304.8 mm.), with a piston of equal area. With an adjustable dam, the height of discharge may be varied from 3 to 4.5 in. (76.2 to 114.3 mm.). In order to study the behavior of the ore-column and bedding during the process of jigging, one side of the jig-box was made of plate glass. Three types of strokes were employed: (1) The eccentric, adjustable within the limits of 0 and 2 in. (0 and 50.8 mm.). (2) Circular-arc cams, where the period of pulsion occupies three-fourths of the revolution of the cam, or eccentric shaft, and suction one-fourth; or by reversing the direction of rotation of the cam-shaft, or slipping the hub and cam off the shaft and turning it end for end, the times or periods are reversed respectively for pulsion and suction. Cams were made having throws up to 2 in. (50.8 mm.), but only the three shortest throws were used—namely, 1 in. (25.4 mm.), 0.5 in. (12.7 mm.), and 0.25 in. (6.35 mm.). (3) Involute cams, in which the periods were divided into thirds, *i.e.*, one-third of the revolution of the cam-shaft devoted to pulsion, and two-thirds to suction; or as noted above, by reversing the direction of rotation of the cam these periods were reversed. All cams were made of wood, and quickly and easily attached to a cast-iron hub, and by means of a set-screw fastened to the shaft, as shown in full in Fig. 4. Circular-arc and involute cams indicate the character of the curves. The circular-arc cams do not give a uniform motion; or in other words, the cam in describing equal arcs in either the pulsion- or suction-period does not cause the piston to travel equal distances. In the involute cams, however, in either pulsion- or suction-periods, equal arcs give equal distances for piston-travel. The manner of communicating motion from the cams or eccentric is clearly indicated in Figs. 1 and 2. These engage with a brass roller attached to a wrought-iron yoke moving between vertical guides. In order to steady and support the yoke still more, a steel rod is attached to the upper end, passing through a hole in a cross-beam, and is attached to the lower end of the yoke of the piston-rod. The roller, yoke, and piston are actuated

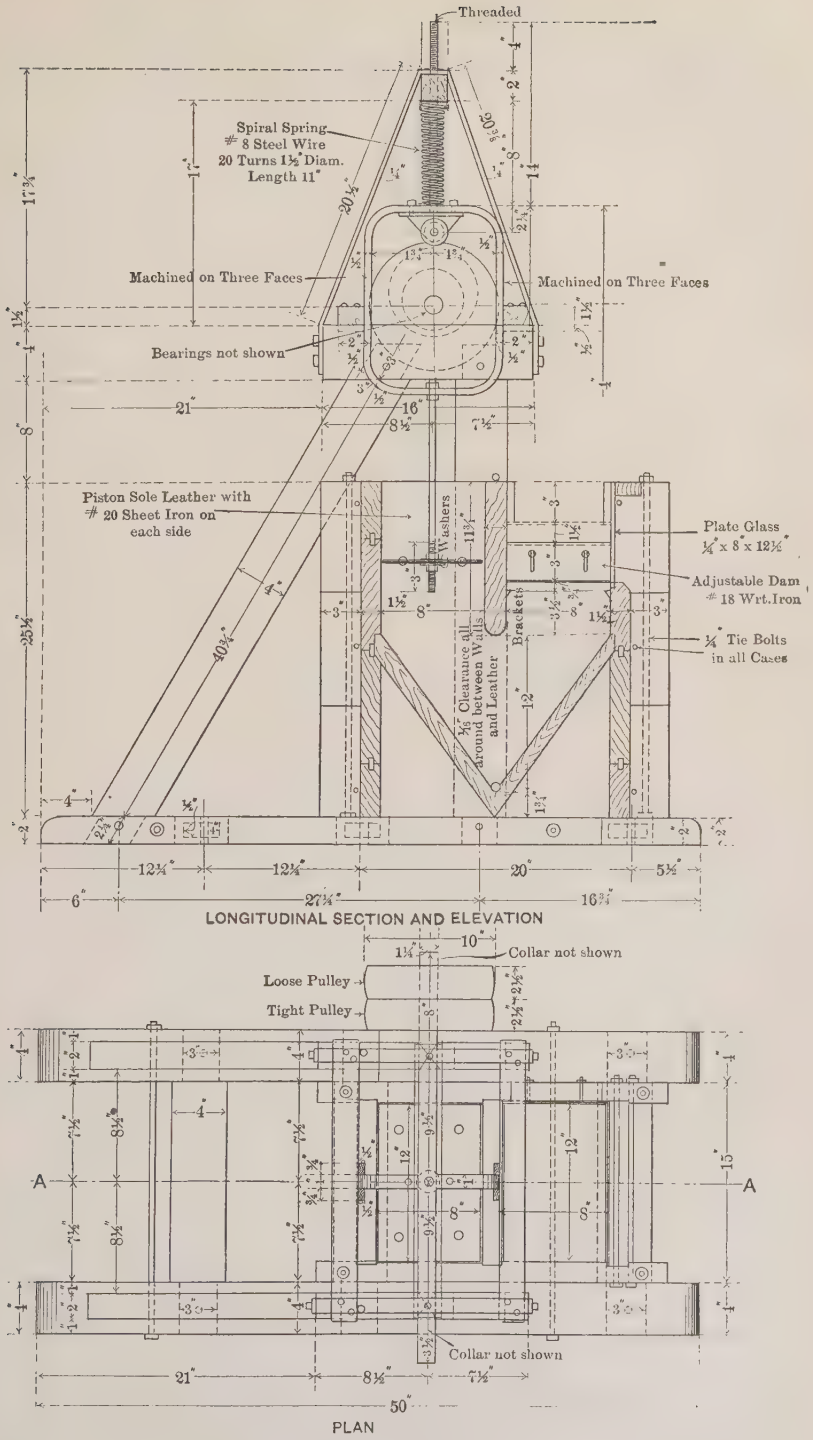
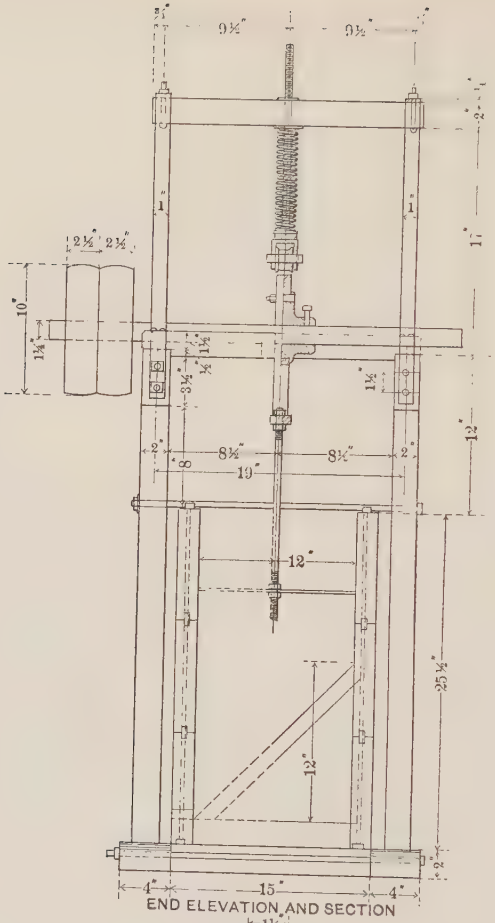
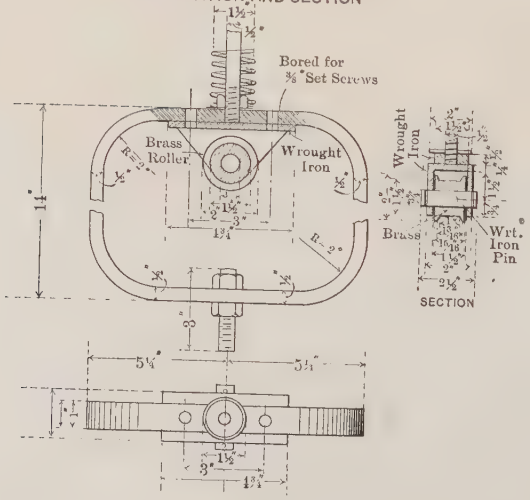


FIG. 1.—THE JARVIS LABORATORY-JIG, LONGITUDINAL SECTION AND ELEVATION, AND PLAN.



END ELEVATION AND SECTION



DETAILS OF WROUGHT IRON YOKE, ROLLER AND BRACKET

FIG. 2.—THE JARVIS LABORATORY-JIG, END ELEVATION AND SECTION, AND DETAILS OF YOKE, ETC.

positively by the cam on the up-stroke, and to secure a strong and quick down-stroke, a spring of 60 lb. pressure per linear inch of compression was employed. This elastic pressure insured a uniform contact of the roller and cam. It is evident that a large number of styles of cam-curves may be used

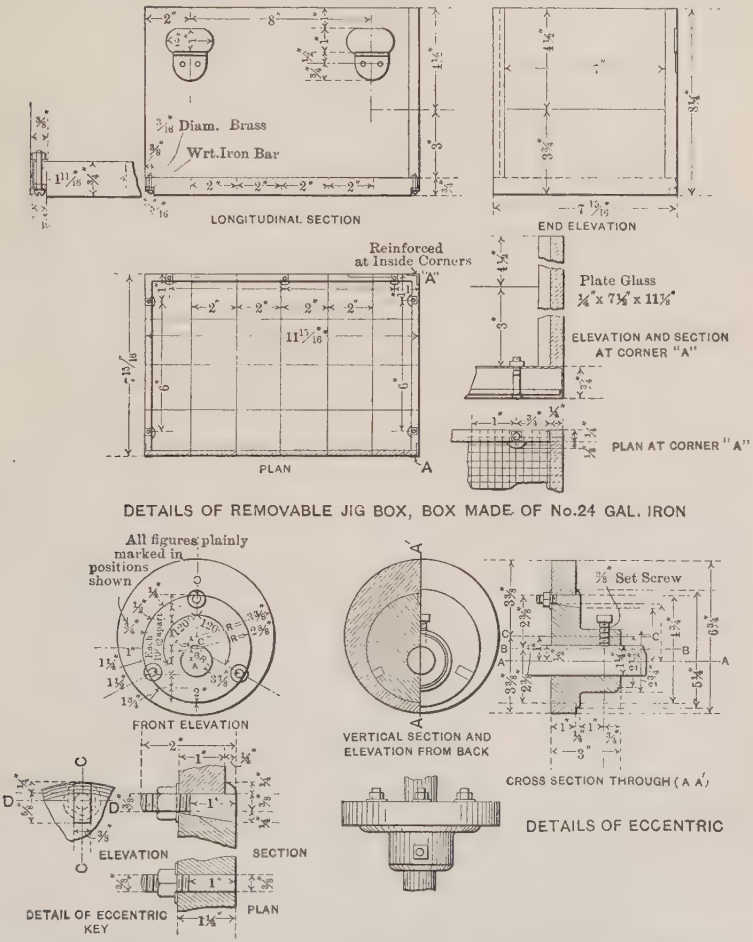


FIG. 3.—THE JARVIS LABORATORY-JIG, DETAILS OF JIG-BOX AND ECCENTRIC.

with this device, and the period of movement of the piston may be varied almost infinitely. It is to be observed, also, that in this system the piston, in all positions, is perfectly horizontal. The piston is made of a single piece of sole-leather, securely riveted between two heavy plates of galvanized iron. With these materials the piston can be run with very little

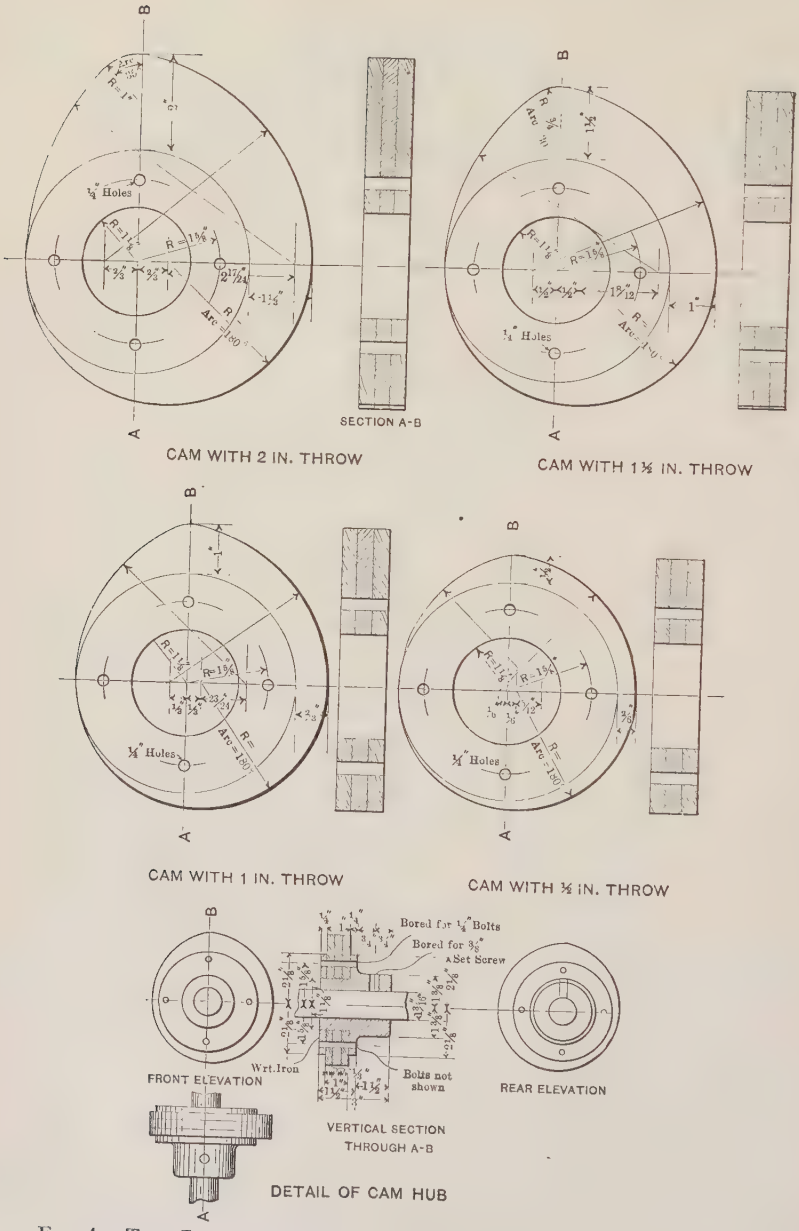


FIG. 4.—THE JARVIS LABORATORY-JIG, ELEVATIONS AND SECTIONS OF CIRCULAR-ARC WOODEN CAMS.

clearance, and there is no danger of warping, swelling, or getting out of repair very easily. The hutch-box sloped from three

sides, at an angle exceeding 50°, to a single spigot in one side of the jig. It was found that at this angle little or no hutch-work collected on the sides, and its entire removal was easily effected. The jig was driven by a 1-h.p. electric motor through the variable-speed counter-shaft. The sieve was supported in a galvanized-iron skeleton, which was removable from the jig-box itself, and different sized screens could readily be inter-

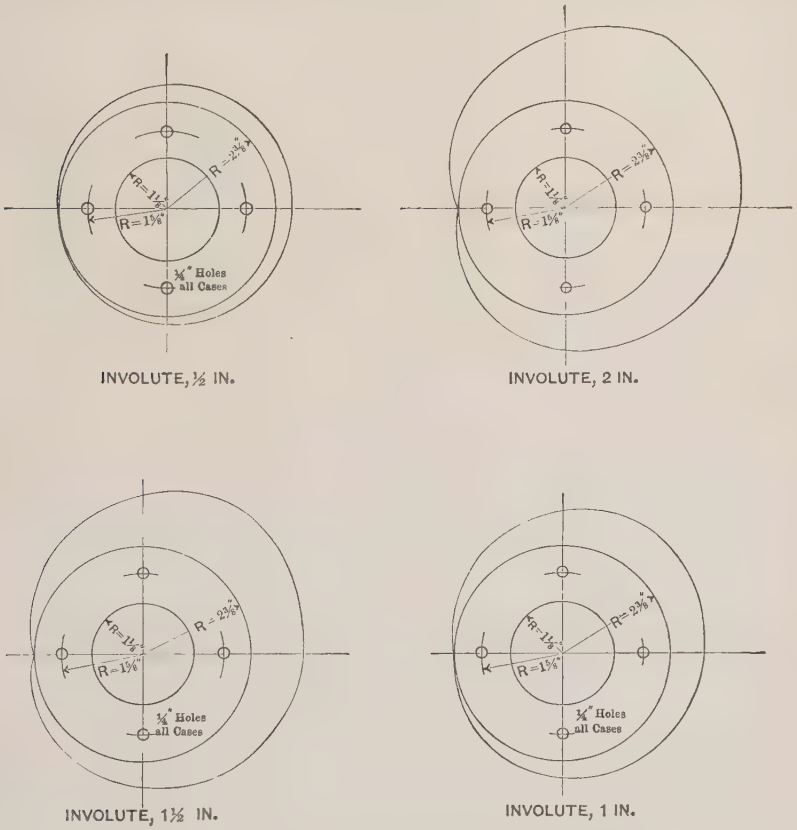


FIG. 5.—THE JARVIS LABORATORY-JIG, CURVES OF INVOLUTE CAMS.

changed. In the tests hereinafter described, only one size sieve—an 8-mesh one—was used.

2. *Materials and Other Accessories.*

A. *Screens.*—Table II. gives the number and mesh of the screen, and the size of the aperture in inches and millimeters. In all cases the holes were square. The size of the hole in the first five sizes was determined by measuring the wire with a wire-

gauge, and counting the number of meshes in a given length. For the remaining screens the diameter of hole was determined by measuring the diameter of the wire and the aperture with a microscopic micrometer, each value given being the mean of several determinations.

B. *Sieve-Sizes.*—The data pertaining to the sieve-sizes are given in Table II.

TABLE II.—*Sieve-Sizes.*

No.	Mesh.	Kind.	Size of Aperture.	
			Inch.	Mm.
1	4	Brass.	0.2097	5.326
2	6	Brass.	0.1382	3.510
3	8	Brass.	0.0966	2.453
4	10	Steel.	0.0841	2.136
5	12	Brass.	0.0654	1.661
6	20	Brass.	0.0381	0.970
7	40	Brass.	0.0165	0.420
8	60	Brass.	0.0102	0.260
9	80	Brass.	0.0082	0.210
10	100	Brass.	0.0063	0.160
		Jig-Sieve.		
1	8	Steel.	0.097	2.464

C. *Bedding.*—The bedding used in all the following tests was sized between the limits of 3.510 and 5.326 mm., or through the 4-mesh sieve and on the 6-mesh sieve, and was maintained at the same thickness, 1.5 in. (38.1 mm.), upon the jig-sieve throughout the experiments.

D. *Minerals.*—The three minerals used were fairly pure. The quartz was kindly furnished by Professor Munroe, and the sphalerite and galena by the Foote Mineral Co., of Philadelphia, Pa.

E. *Specific Gravities.*—The specific gravity of each mineral was: galena, 6.66; sphalerite, 3.74; and quartz, 2.62.

The low specific gravity of the metallic minerals used indicates that they are not pure, and an examination revealed the presence of included quartz and minute quantities of other minerals. In crushing these minerals, all the quartz particles that could be picked out by hand were removed. The values given, however, are those obtained for the crushed minerals, ready to be added to the feed.

These three minerals were selected since zinc-blende and galena represent about the minimum and maximum limits

respectively of the ores usually treated on jigs. In thus examining the two limits, the behavior of intermediate minerals could be closely predicted.

F. *Feed.*—The feed in all the tests was crushed by stages until small enough to pass the 10-mesh (2.136-mm.) screen. This size represented the maximum, from which it varied to that of the finest dust. Two classes of feed were employed. The first contained 10 per cent., by weight, of heavy mineral (galena or blende), and the second 20 per cent. of heavy mineral. The balance was, respectively, 90 or 80 per cent. of quartz. Table III. shows the screen-analysis of the three minerals constituting the feed.

TABLE III.—*Screen-Analysis of Minerals in Feed.*

All through 10-mesh (2.136 mm.), and on, mesh, . . .	12.	20.	40.	60.	80.	100.	Through 100.	
On, mm., ...	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
Mineral.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Total.
Galena, ...	9.6	23.1	26.3	10.8	5.4	2.8	21.2	99.2
Sphalerite, ...	10.5	24.5	26.8	10.5	5.4	2.9	18.6	99.2
Quartz, ...	17.1	29.1	26.4	9.5	4.5	2.3	11.0	99.9

The values in Table III. represent the mean of three or four different determinations, made after crushing a large lot and thoroughly sampling it down.

Table IV. shows the calculated percentages of galena and quartz in the two classes of feed, based upon the screen-analysis of the pure minerals given in Table III.

TABLE IV.—*Analyses of Ten- and Twenty-Per Cent. Galena Feed.*

	Ten-Per Cent. Galena Feed.							
Mesh. { inch / mm.	12. / 1.66	20. / 0.97	40. / 0.42	60. / 0.26	80. / 0.21	100. / 0.16	Thro' 100. / 0.16	Average.
	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.
Quartz.....	94.1	92.0	90.0	88.8	88.2	88.1	82.3	89.1
Galena.....	5.9	8.0	10.0	11.2	11.8	11.9	17.7	10.9
	Twenty-Per Cent. Galena Feed.							
	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.
Quartz.....	87.7	83.4	80.0	77.7	77.3	76.8	67.7	78.7
Galena.....	12.3	16.6	20.0	22.7	22.3	23.2	32.3	21.3

The results obtained for sphalerite and quartz are given in Table V.

TABLE V.—*Analyses of Ten- and Twenty-Per Cent. Sphalerite Feed.*

Mesh. inch / mm.	12. / 1.66	20. / 0.97	40. / 0.42	60. / 0.26	80. / 0.21	100. / 0.16	Thro' 100. / 0.16	Average.
Ten-Per Cent. Sphalerite Feed.								
	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.
Quartz	93.6	91.5	90.0	89.2	88.3	87.7	84.6	89.3
Sphalerite	6.4	8.5	10.0	10.8	11.7	12.3	15.4	10.7
Twenty-Per Cent. Sphalerite Feed.								
	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.
Quartz	86.7	82.7	80.0	78.3	77.0	75.9	70.4	78.7
Sphalerite	13.3	17.3	20.0	21.7	23.0	24.1	29.6	21.3

In Tables IV. and V. the columns for each of the respective feeds show the percentages of each of the two minerals on the different screen-sizes. Thus, in Table IV., with 10 per cent. of galena, the material resting on the 12-mesh (1.66-mm.) sieve contained 94.1 per cent. of quartz and 5.9 per cent. of galena, etc.

With both sphalerite and galena, the screen-analyses, and from these the calculated percentages of the mineral-content of each screen-size, show that more fine material is produced in crushing these softer minerals than in crushing quartz. The finest size of the 10 and the 20 per cent. galena or sphalerite shows a much higher percentage of these minerals than the average of the feed, as shown in Tables IV. and V.

3. *Method of Conducting the Tests.*

In beginning a series of tests on a given feed, the exact proportion of each mineral was weighed out, so that the total quantity was 35 lb. (15.87 kg.). Meanwhile, the sieve had received its bedding, 1.5 in. (38.1 mm.), and the hutch-box and jig were filled with water; the tailings-trough placed in position, connecting with a large tub in which all the overflow and tailings were caught; the feed thoroughly wetted down (if fresh material); power was turned on and the jig started. In case it was the first run of a series, the jig-box containing bedding only, the feed was rapid until this was filled with the mixture, after which the feeding proceeded at the regular rate.

The feeding was accomplished by filling with the ore-mixture a large flat-bottomed scoop, of a width slightly less than that of the jig-compartment, 8 in. (203.2 mm.), and with a small and constant stream of water washing the material from the scoop on to the jig. While the speed of jigging and the rate of feeding varied, the object always aimed at was to feed the jig just as fast as it appeared able to treat the material. The discharge was watched constantly to see if any particles of heavy mineral were being carried into the tailings. If so, the rate of feeding was reduced. Close watch was also kept on the jig-bed, and if the jig showed symptoms of clogging up, due to rapid feeding, the rate of feed was immediately decreased.

At the end of the run, usually from 8 to 15 min., the jig was stopped, the water-supply cut off, and the hutch-products drawn off into suitable vessels. After allowing the material to settle, the water was carefully decanted and the products thoroughly mixed, and a sample of about 125 g. cut out, which was dried, and later exactly 100 g. of this sample was weighed out on a pulp-balance and sized on a nest of sieves, ranging from 12-mesh (1.66 mm.) through 100-mesh (0.16 mm.), and each size carefully weighed; finally, the percentage of galena or sphalerite in each sieve-size was determined. The analyses of the products were made in several ways. In the first two or three coarse sizes good results were obtained by weighing out 1 or 2 g. and picking out the quartz or other mineral by hand and then weighing again; also, by comparing with standard mixtures of quartz and galena or sphalerite. In the small sizes vanning-tests were made.

After the completion of a run, the tailings, which were given ample time in which to allow the fine material to settle and the water to be decanted off, were again mixed with the product from the hutch and formed the feed for another test. The material was thus used repeatedly until all the tests had been completed for a particular series or class. The material remaining in the jig-box was not cleaned out from test to test, unless another feed was to be employed. The investigations had to do only with what passed into the hutch, and determinations upon the character and nature of what remained on the sieve, except as it could be examined through the glass side of the jig, were not made.

4. *Record of Results.*

In the following records are five horizontal rows of figures: in the topmost row, the sieve-mesh; in the next lower row, the corresponding size in millimeters of the aperture upon which the material was caught; and three lower rows marked "*A*," "*B*," and "*C*" respectively. The first of these, *A*, gives the weights in grams of the different sieve-sizes; and since these are all on a basis of 100 g. the weights, therefore, represent percentages as well. Row *B* gives the percentage of heavy mineral, galena or sphalerite, in each of the sieve-products, and the balance in every case is quartz. Row *C* gives the weight of heavy mineral contained in each of the sieve-sizes, and is obtained by multiplying the weights in row *A* by the respective percentages in the *B* row. The sum of the products in the *C* row gives the number of grams of mineral in 100 g. of the concentrate, or in other words, the percentage.

Under the stroke of each experiment are given: (1), the number of revolutions of the cam or eccentric shaft per minute; (2) the length in inches and millimeters; (3) the kind of stroke; (4) pulsion, in which the fractions $\frac{1}{4}$, $\frac{1}{3}$, $\frac{1}{2}$, $\frac{2}{3}$, and $\frac{3}{4}$ refer to the fractional part of the entire revolution of the cam or shaft in which this movement took place. The smaller this fraction the quicker the movement. The rates or velocities are set opposite. The same is true for the period of suction.

The observed pulsion- and suction-velocities noted in the following tests and elsewhere in this paper are to be understood as the mean piston-velocities, or the velocities of the water-column in the free part of the jig-column, and not the actual current-velocities acting upon a mass of grains constituting the jig-bed.

In studying these experiments, reference should be made to Figs. 6 to 13, inclusive, in which row *C* is shown graphically, representing the mean diameter of grains.

TEST 1.—*Galena* 10, *Quartz* 90 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, 1 in. (25.4 mm.).
Kind: Circular-arc cam.
Pulsion: $\frac{1}{4}$ (270.7 mm.) = 10.66 in. per sec.
Suction: $\frac{3}{4}$ (90.2 mm.) = 3.55 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100.	thro' 100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	3.3	11.6	36.0	18.4	9.5	4.7	16.0	99.5
B.	100.0	80.0	35.0	20.0	15.0	20.0	28.0	
C.	3.3	9.3	12.6	3.7	1.4	0.9	4.5	35.7

Percentage of galena in concentrates: 35.7.

Ratio of concentration based on original feed: 3.57.

Remarks.—All the material on the jig-bed pulsated—the material above having a longer amplitude than the grains deeper down. It was observed that the bedding-grains at the top moved nearly 0.75 in. vertically, while those at the bottom of the bed next to the screen moved about 0.25 in.

TEST 2.—*Galena* 10, *Quartz* 90 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, 1 in. (25.4 mm.).
Kind: Circular-arc cam.
Pulsion: $\frac{3}{4}$ (90.2 mm.) = 3.55 in. per sec.
Suction: $\frac{1}{4}$ (270.7 mm.) = 10.66 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100. thro'	100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	0.7	7.0	26.5	22.2	14.0	6.6	22.3	99.3
B.	100.0	95.0	60.0	35.0	25.0	20.0	30.0	
C.	0.7	6.6	15.9	7.7	3.5	1.3	6.7	42.4

Percentage of galena in concentrates: 42.4.

Ratio of concentration based on original feed: 4.24.

Remarks.—Some movement of the bedding-grains, especially near the top, but only a few grains of galena were visible in the interstitial spaces of the bedding. The ore-column pulsated violently, and between the bedding and the ore was a zone in very active motion, while above the column of ore was quite compact.

TEST 3.—*Galena* 10, *Quartz* 90 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, 1 in. (25.4 mm.).
Kind: Involute cam.
Pulsion: $\frac{1}{3}$ (203.2 mm.) = 8 in. per sec.
Suction: $\frac{2}{3}$ (101.6 mm.) = 4 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100. thro'	100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	2.4	12.7	35.6	18.0	10.3	5.6	14.7	99.3
B.	95.0	75.0	25.0	16.0	15.0	15.0	20.0	
C.	2.3	10.5	8.9	3.0	1.5	0.8	2.9	29.9

Percentage of galena in concentrates: 29.9.

Ratio of concentration based on original feed: 2.99.

Remarks.—The bedding and with it the ore-column pulsated. The grains of bedding were kept in constant circulation. Very few grains of galena could be seen in the interstitial spaces of the bedding, but were free. It was evident, therefore, that if a galena- or quartz-grain got as far as the bedding it had little opportunity of remaining there.

TEST 4.—*Galena* 10, *Quartz* 90 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, 1 in. (25.4 mm.).
Kind: Involute cam.
Pulsion: $\frac{2}{3}$ (101.6 mm.) = 4 in. per sec.
Suction: $\frac{1}{3}$ (203.2 mm.) = 8 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100. thro'	100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	0.4	7.8	39.7	21.4	10.7	4.7	14.8	99.5
B.	60.0	50.0	25.0	22.0	18.0	18.0	22.0	
C.	0.2	3.9	9.9	4.7	1.9	0.8	3.2	24.6

Percentage of galena in concentrates: 24.6.

Ratio of concentration based on original feed: 2.46.

Remarks.—The ore-bed pulsated violently, but not so much so as with the strong suction of the circular-arc cam in Test 3. The grains of the bed did not behave exactly alike, and the middle of the bed contained some grains of galena.

TEST 5.—*Galena* 10, *Quartz* 90 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, 1 in. (25.4 mm.).
Kind: Eccentric.
Pulsion and Suction: ½ (135.5 mm.) = 5.33 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	1.4	9.0	34.0	20.5	11.7	6.0	17.2	99.8
B.	95.0	95.0	35.0	20.0	20.0	20.0	28.0	
C.	1.3	8.5	11.9	4.1	2.3	1.2	4.8	34.1

Percentage of galena in concentrates: 34.1.

Ratio of concentration based on original feed: 3.41.

Remarks.—The entire bed pulsated much more uniformly than in Tests 3 and 4. The bedding-grains were free to move, and tended to move in convection-currents. No particles of galena collected on top of the bedding, and few could be seen in the interstitial spaces.

TEST 6.—*Galena* 10, *Quartz* 90 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, ½ in. (12.7 mm.).
Kind: Eccentric.
Pulsion and Suction: ½ (67.7 mm.) = 2.66 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	.....	2.7	26.8	20.7	13.2	10.0	26.2	99.6
B.	.....	95.0	78.0	48.0	33.0	27.0	38.0	
C.	.....	2.5	20.9	9.9	4.3	2.7	9.9	50.2

Percentage of galena in concentrates: 50.2.

Ratio of concentration based on original feed: 5.02.

Remarks.—The lower third of bedding quite fixed, while the upper two-thirds pulsated, but the grains did not change positions—moving *en masse*. The ore-column pulsated regularly, and between the bedding and the ore was a zone of great mobility. The action and movement going on in the ore-column resembled very much that taking place in a hydraulic classifier.

TEST 7.—*Galena* 10, *Quartz* 90 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, ½ in. (12.7 mm.).
Kind: Circular-arc cam.
Pulsion: ¼ (135.5 mm.) = 5.33 in. per sec.
Suction: ¾ (45.2 mm.) = 1.77 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	0.8	6.4	24.7	21.4	13.2	7.5	26.0	100.0
B.	100.0	97.0	60.0	35.0	18.0	20.0	27.0	
C.	0.8	6.2	14.8	7.5	2.3	1.5	7.0	40.1

Percentage of galena in concentrates : 40.1.

Ratio of concentration based on original feed : 4.01.

Remarks.—The bedding and the ore-column pulsated uniformly—the top having a longer amplitude and extending over a longer time than the grains nearer the bottom. Ore-column very mobile and in active circulation. The upper third of bedding contained many particles of galena.

TEST 8.—*Galena* 10, *Quartz* 90 *per cent.*

Stroke : Cam-shaft rev. per min., 160. Length, $\frac{1}{2}$ in. (12.7 mm.).

Kind : Circular-arc cam.

Pulsion : $\frac{3}{4}$ (45.2 mm.) = 1.77 in. per sec.

Suction : $\frac{1}{4}$ (135.5 mm.) = 5.33 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100. thro' 100.		Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.		2.1	26.0	24.4	14.7	7.1	25.3	99.6
B.		90.0	53.0	31.0	20.0	20.0	27.0	
C.		1.9	13.8	7.5	2.9	1.4	6.8	34.3

Percentage of galena in concentrates : 34.3.

Ratio of concentration based on original feed : 3.43.

Remarks.—Only the top third of bedding showed any signs of movement, but the interstitial spaces were filled with particles of galena. The particles in the ore-column tended to circulate in two opposite and distinct paths.

TEST 9.—*Galena* 10, *Quartz* 90 *per cent.*

Stroke : Cam-shaft rev. per min., 160. Length, $\frac{1}{2}$ in. (12.7 mm.).

Kind : Involute cam.

Pulsion : $\frac{1}{3}$ (101.6 mm.) = 4 in. per sec.

Suction : $\frac{2}{3}$ (50.8 mm.) = 2 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100. thro' 100.		Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	1.3	10.3	27.9	20.5	11.9	6.4	21.2	99.5
B.	100.0	100.0	72.0	31.0	22.0	20.0	31.0	
C.	1.3	10.3	20.0	6.3	2.6	1.3	6.5	48.3

Percentage of galena in concentrates : 48.3.

Ratio of concentration based on original feed : 4.83.

Remarks.—Both the bedding and the ore-column pulsated—the top having a longer amplitude of vibration and requiring a longer time than the grains nearer the bottom. Many grains of galena in the upper third of bedding and decreasing below. The ore-column very mobile, and line between bedding and ore horizontal and uniformly even.

TEST 10.—*Galena* 10, *Quartz* 90 *per cent.*

Stroke : Cam-shaft rev. per min., 160. Length, $\frac{1}{2}$ in. (12.7 mm.).

Kind : Involute cam.

Pulsion : $\frac{2}{3}$ (50.8 mm.) = 2 in. per sec.

Suction : $\frac{1}{3}$ (101.6 mm.) = 4 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100. thro' 100.		Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.		1.0	26.4	25.1	13.7	7.2	26.5	99.9
B.		90.0	70.0	30.0	28.0	26.0	32.0	
C.		0.9	20.5	7.5	3.8	1.8	8.5	43.0

Percentage of galena in concentrates: 43.0.

Ratio of concentration based on original feed: 4.30.

Remarks.—The bedding-grains were practically stationary—neither pulsation nor movement among themselves, and were filled with particles of galena. The ore-column pulsated, but was not mobile except for a zone 0.5 in. thick on top of the bedding. Evidently too much suction.

TEST 11.—*Galena* 10, *Quartz* 90 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, ¼ in. (6.35 mm.).
Kind: Involute cam.
Pulsion: ⅓ (50.8 mm.) = 2 in. per sec.
Suction: ⅔ (25.4 mm.) = 1 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100. thro'	100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.		2.4	24.1	20.0	16.3	9.4	27.8	100.0
B.		95.0	92.0	56.0	35.0	22.0	25.0	
C.		2.3	22.1	11.2	5.7	2.0	5.7	49.0

Percentage of galena in concentrates: 49.0.

Ratio of concentration based on original feed: 4.90.

Remarks.—The bedding, as a whole, did not pulsate, but the grains in the upper part of the bedding showed some movement, and this portion was filled with particles of galena. The ore-column was very mobile and pulsated regularly and uniformly. The large grains of quartz rested directly upon the bedding, with the finer quartz-particles above.

TEST 12.—*Galena* 10, *Quartz* 90 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, ¼ in. (6.35 mm.).
Kind: Involute cam.
Pulsion: ⅔ (25.4 mm.) = 1 in. per sec.
Suction: ⅓ (50.8 mm.) = 2 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100. thro'	100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.		1.4	20.4	24.5	18.8	9.1	25.5	99.7
B.		80.0	62.0	40.0	27.0	24.0	25.0	
C.		1.1	12.6	9.8	5.0	2.2	6.3	37.0

Percentage of galena in concentrates: 37.0.

Ratio of concentration based on original feed: 3.7.

Remarks.—The bedding did not move at all. The ore-bed seemed to be quite mobile immediately above the bedding, but compact close to the top.

TEST 13.—*Galena* 10, *Quartz* 90 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, ¼ in. (6.35 mm.).
Kind: Circular-arc cam.
Pulsion: ¼ (67.7 mm.) = 2.66 in. per sec.
Suction: ¾ (22.5 mm.) = 0.88 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100. thro'	100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.		4.2	24.8	25.5	15.1	7.3	23.1	100.0
B.		95.0	62.0	26.0	20.0	18.0	27.0	
C.		4.0	15.3	6.6	3.0	1.3	6.3	36.5

Percentage of galena in concentrates: 36.5.

Ratio of concentration based on original feed: 3.65.

Remarks.—The bedding and the ore-column pulsated, but the grains of bedding were not sufficiently mobile to rearrange themselves, although the upper third was much more mobile and pulsated much more than the bottom, and many particles of galena were contained in the interstitial spaces of the upper third of bedding. The entire ore-column was very free and mobile and pulsated uniformly.

TEST 14.—*Galena* 10, *Quartz* 90 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, $\frac{1}{4}$ in. (6.35 mm.).
Kind: Circular-arc cam.
Pulsion: $\frac{3}{4}$ (22.5 mm.) = 0.88 in. per sec.
Suction: $\frac{1}{4}$ (67.7 mm.) = 2.66 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100. thro'100.		Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	.....	2.0	24.6	22.6	14.4	7.0	29.4	100.0
B.	.....	94.0	62.0	25.0	20.0	22.0	33.0	
C.	.....	2.0	15.2	4.6	2.9	1.5	9.7	35.9

Percentage of galena in concentrates: 35.9.

Ratio of concentration based on original feed: 3.59.

Remarks.—No movement in the bedding, although the top bedding-grains showed some tendency to move, and many particles of galena could be seen in the upper third of the bedding. The ore-column pulsated regularly, and was quite compact

TEST 15.—*Galena* 10, *Quartz* 90 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, $\frac{1}{4}$ in. (6.35 mm.).
Kind: Eccentric.
Pulsion and Suction: $\frac{1}{2}$ (33.9 mm.) = 1.33 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100. thro'100.		Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	.....	0.4	17.8	23.9	17.2	9.3	31.0	99.6
B.	.....	85.0	63.0	31.0	20.0	20.0	30.0	
C.	.....	0.3	11.2	7.4	3.4	1.8	9.3	33.4

Percentage of galena in concentrates: 33.4.

Ratio of concentration based on original feed: 3.34.

Remarks.—The bedding was quite fixed in position, and the upper part well filled with grains of galena. It was noticed that when the feed was too fast, an inclined line, beginning at the top of the bedding at the back of the jig-box, and sloping up nearly to the top of the ore-column at the front or discharge was formed. Otherwise the ore-column was mobile, with the coarse particles of quartz resting above the bedding, and the finer particles arranged above.

TEST 16.—*Galena* 10, *Quartz* 90 *per cent.*

Stroke: Cam-shaft rev. per min., 320. Length, $\frac{1}{4}$ in. (6.35 mm.).
Kind: Eccentric.
Pulsion and Suction: $\frac{1}{2}$ (67.7 mm.) = 2.66 in. per sec.

On mesh.....	12.	20.	40.	60.	80	100. thro'100.		Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	0.8	6.4	31.5	26.9	11.9	5.8	16.4	99.7
B.	100.0	98.0	60.0	25.0	15.0	20.0	30.0	
C.	0.8	6.4	18.9	6.7	1.8	1.1	4.9	40.6

Percentage of galena in concentrates : 40.6.

Ratio of concentration based on original feed : 4.06.

Remarks.—Both the bedding and the ore-column pulsated regularly—the grains near the top of the bedding having a longer amplitude of vibration than those near the bottom, and the same being true of the grains in the ore-column. The ore-column was very mobile. The jig worked fast.

Test 17.—*Galena* 10, *Quartz* 90 *per cent.*

Stroke : Cam-shaft rev. per min., 320. Length, $\frac{1}{8}$ in. (3.17 mm.).

Kind : Eccentric.

Pulsion and Suction : $\frac{1}{2}$ (33.9 mm.) = 1.33 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100.	thro' 100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	.....	4.2	31.3	27.2	13.5	6.4	17.4	100.0
B.	.....	95.0	35.0	21.0	18.0	18.0	28.0	
C.	.....	4.0	10.9	5.7	2.4	1.1	4.9	29.0

Percentage of galena in concentrates : 29.0

Ratio of concentration based on original feed : 2.90.

Remarks.—The entire bed pulsated, and the bedding contained many particles of galena and some quartz. As noted before, the top had a longer amplitude of vibration than the bottom, and required a longer time. The ore-column pulsated regularly, and the fine material (quartz) was carried down to the bedding so that it was distributed quite regularly throughout the ore. The ore-column was compact.

Test 18.—*Galena* 10, *Quartz* 90 *per cent.*

Stroke : Cam-shaft rev. per min., 400. Length, $\frac{1}{16}$ in. (1.59 mm.).

Kind : Eccentric.

Pulsion and Suction : $\frac{1}{2}$ (21.2 mm.) = 0.83 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100.	thro' 100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	.....	2.6	37.3	26.0	12.1	5.8	16.1	99.9
B.	.....	100.0	25.0	23.0	16.0	15.0	26.0	
C.	.....	2.6	9.3	6.0	1.9	0.8	4.2	24.8

Percentage of galena in concentrates : 24.8.

Ratio of concentration based on original feed : 2.48.

Remarks.—The grains of the bedding were not very mobile, and only the top layer of grains showed any indication of pulsating. The base of the ore-column was distinguished by a zone of active agitation. Above this zone, which was only 0.5 in. thick, the ore-column was compact and not mobile. The bedding-grains contained only a few galena-grains in the upper third portion, but the interstitial spaces were filled with quartz. In the middle and lower third portions of the bedding, many more grains of galena were visible, being more abundant in the middle third.

TEST 21.—*Galena* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, 1 in. (25.4 mm.).
Kind: Circular-arc cam.
Pulsion: $\frac{1}{4}$ (270.7 mm.) = 10.66 in. per sec.
Suction: $\frac{3}{4}$ (90.2 mm.) = 3.55 in. per sec.

On mesh..... / Size in mm..	12. / 1.66	20. / 0.97	40. / 0.42	60. / 0.26	80. / 0.21	100. / 0.16	thro'100. / 0.16	Total.
A.	4.3	18.1	33.2	16.7	8.5	4.0	14.8	99.6
B.	100.0	90.0	55.0	50.0	40.0	40.0	50.0	
C.	4.3	16.2	18.1	8.5	3.4	1.6	7.5	59.6

Percentage of galena in concentrates: 59.6.
Ratio of concentration based on original feed: 2.98.
Remarks.—Movement of jig-bed same as Test 1.

TEST 22.—*Galena* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, 1 in. (25.4 mm.).
Kind: Circular-arc cam.
Pulsion: $\frac{3}{4}$ (90.2 mm) = 3.55 in. per sec.
Suction: $\frac{1}{4}$ (270.7 mm.) = 10.66 in. per sec.

On mesh..... / Size in mm..	12. / 1.66	20. / 0.97	40. / 0.42	60. / 0.26	80. / 0.21	100. / 0.16	thro'100. / 0.16	Total.
A.	0.5	6.6	30.4	19.6	11.5	5.6	25.7	99.9
B.	100.0	90.0	65.0	55.0	45.0	45.0	50.0	
C.	0.5	5.9	19.5	11.0	5.1	2.5	12.8	57.3

Percentage of galena in concentrates: 57.3.
Ratio of concentration based on original feed: 2.86.
Remarks.—Movement of jig-bed similar to Test 2.

TEST 23.—*Galena* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, 1 in. (25.4 mm.).
Kind: Involute cam.
Pulsion: $\frac{1}{3}$ (203.2 mm.) = 8 in. per sec.
Suction: $\frac{2}{3}$ (101.6 mm.) = 4 in. per sec.

On mesh..... / Size in mm..	12. / 1.66	20. / 0.97	40. / 0.42	60. / 0.26	80. / 0.21	100. / 0.16	thro'100. / 0.16	Total.
A.	3.4	16.5	32.4	18.0	9.2	4.3	15.6	99.4
B.	100.0	95.0	55.0	40.0	40.0	40.0	45.0	
C.	3.4	15.6	17.6	7.2	3.6	1.6	7.0	56.0

Percentage of galena in concentrates: 56.0.
Ratio of concentration based on original feed: 2.80.
Remarks.—Movement of bed similar to Test 3.

TEST 24.—*Galena* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, 1 in. (25.4 mm.).
Kind: Involute cam.
Pulsion: $\frac{2}{3}$ (101.6 mm.) = 4 in. per sec.
Suction: $\frac{1}{3}$ (203.2 mm.) = 8 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100. thro' 100.		Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	1.5	12.2	34.0	20.0	10.2	4.8	17.5	100.2
B.	95.0	80.0	56.0	45.0	40.0	45.0	45.0	
C.	1.4	9.6	18.7	9.0	4.0	2.1	7.7	52.5

Percentage of galena in concentrates: 52.5.
Ratio of concentration based on original feed: 2.62.
Remarks.—The entire bed pulsated, but not so violently as Test 23. The ore-column pulsated much more than the bedding, and the top of the bedding than the bottom. Between the bedding and the ore-column was a zone 0.5 in. thick of great activity. Few grains in the interstitial spaces of the bedding. Jigged rapidly.

TEST 25.—*Galena* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, 1 in. (25.4 mm.).
Kind: Eccentric.
Pulsion and Suction: $\frac{1}{2}$ (135.5 mm.) = 5.33 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100. thro' 100.		Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	4.6	17.4	28.8	17.1	9.5	5.5	17.2	100.1
B.	100.0	95.0	80.0	50.0	40.0	50.0	45.0	
C.	4.6	17.0	23.7	8.5	3.8	2.2	7.6	67.4

Percentage of galena in concentrates: 67.4.
Ratio of concentration based on original feed: 3.37.
Remarks.—The entire bed pulsated, the upper part having a longer amplitude of vibration and requiring a longer time to complete it than the grains nearer the bottom. Difficult to save the finest grains of galena. The bedding-grains were free to change positions during the pulsion-cycle.

TEST 26.—*Galena* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, $\frac{1}{2}$ in. (12.7 mm.).
Kind: Eccentric.
Pulsion and Suction: $\frac{1}{2}$ (67.7 mm.) = 2.66 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100. thro' 100.		Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	0.7	7.3	26.0	16.9	11.7	7.4	29.5	99.5
B.	100.0	100.0	85.0	60.0	45.0	40.0	40.0	
C.	0.7	7.3	22.1	10.2	5.4	2.8	12.0	60.5

Percentage of galena in concentrates: 60.5
Ratio of concentration based on original feed: 3.02.
Remarks.—The entire bed pulsated, and the zone between the bedding and the

ore-column was an active one—the grains in the ore-column were kept in constant circulation. The interstitial spaces in the upper third of the bedding filled with particles of galena.

TEST 27.—*Galena* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, ½ in. (12.7 mm.).
Kind: Circular-arc cam.
Pulsion: ¼ (135.5 mm.) = 5.33 in. per sec.
Suction: ¾ (45.2 mm.) = 1.77 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	2.4	15.0	27.9	18.4	11.4	6.0	18.7	99.8
B.	100.0	100.0	85.0	50.0	35.0	35.0	35.0	
C.	2.4	15.0	23.8	9.2	3.8	2.1	6.6	62.9

Percentage of galena in concentrates: 62.9.
Ratio of concentration based on original feed: 3.15.
Remarks.—The entire bed pulsated very uniformly, the top having a longer time to complete it than the grains nearer the bottom. Many grains of galena were observed in the upper part of the bedding, but only a few in the lower half.

TEST 28.—*Galena* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, ½ in. (12.7 mm.).
Kind: Circular-arc cam.
Pulsion: ¾ (45.2 mm.) = 1.77 in. per sec.
Suction: ¼ (135.5 mm.) = 5.33 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	0.1	1.7	25.0	22.0	12.7	7.3	31.2	100.0
B.	100.0	95.0	85.0	60.0	45.0	45.0	45.0	
C.	0.1	1.7	21.2	13.2	5.8	3.1	13.9	59.0

Percentage of galena in concentrates: 59.0.
Ratio of concentration based on original feed: 2.95.
Remarks.—The bedding-grains did not pulsate, although those near the top exhibited a slight tendency. The ore-column pulsated, but excepting a zone about 0.5 in. thick immediately above the bedding was otherwise compact. The ore-grains circulated in two distinct and opposite paths.

TEST 29.—*Galena* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, ½ in. (12.7 mm.).
Kind: Involute cam.
Pulsion: ⅓ (101.6 mm.) = 4 in. per sec.
Suction: ⅔ (50.8 mm.) = 2 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	3.4	16.0	25.1	17.2	11.2	6.2	20.7	99.8
B.	100.0	100.0	85.0	60.0	45.0	40.0	45.0	
C.	3.4	16.0	21.2	10.2	4.9	2.4	9.4	67.5

Percentage of galena in concentrates: 67.5.

Ratio of concentration based on original feed: 3.37.

Remarks.—The entire bed pulsated, the upper part, as noted before, having a longer amplitude of vibration and requiring a longer time to complete it than the grains beneath. The upper half of the bedding contained many particles of galena, while only a few were visible in the lower half.

TEST 30.—*Galena* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, $\frac{1}{2}$ in. (12.7 mm.).
Kind: Involute cam.
Pulsion: $\frac{2}{3}$ (50.8 mm.) = 2 in. per sec.
Suction: $\frac{1}{3}$ (101.6 mm.) = 4 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100. thro' 100.		Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	0.1	3.0	31.3	21.6	11.5	6.5	25.7	99.7
B.	100.0	95.0	65.0	50.0	40.0	40.0	45.0	
C.	0.1	2.8	20.1	11.0	4.8	2.6	11.7	53.1

Percentage of galena in concentrates: 53.1.

Ratio of concentration based on original feed: 2.65.

Remarks.—The bedding exhibited a slight tendency to pulsate *en masse*. The upper part of the bedding well filled with particles of galena, decreasing rapidly in number below. Immediately above the bedding the ore-column presented a zone of active agitation about 0.5 in. thick, while above the particles seemed quite compact and not very mobile.

TEST 31.—*Galena* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, $\frac{1}{4}$ in. (6.35 mm.).
Kind: Involute cam.
Pulsion: $\frac{1}{3}$ (50.8 mm.) = 2 in. per sec.
Suction: $\frac{2}{3}$ (25.4 mm.) = 1 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100. thro' 100.		Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	.....	1.4	23.0	17.0	14.6	9.0	35.0	100.0
B.	.....	100.0	100.0	85.0	60.0	50.0	45.0	
C.	.....	1.4	23.0	14.4	8.7	4.5	15.7	67.7

Percentage of galena in concentrates: 67.7.

Ratio of concentration based on original feed: 3.39.

Remarks.—The upper one-third of the bedding-grains exhibited some tendency to arrange themselves during pulsion, but the lower two-thirds did not move or pulsate. In the upper third were many particles of galena and less below. The ore-column pulsated regularly, the large grains of quartz arranging themselves next to the bedding, the smaller on top. The ore-column was mobile.

TEST 32.—*Galena* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, $\frac{1}{4}$ in. (6.35 mm.).
Kind: Involute cam.
Pulsion: $\frac{2}{3}$ (25.4 mm.) = 1 in. per sec.
Suction: $\frac{1}{3}$ (50.8 mm.) = 2 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100. thro'100.		Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.		1.2	17.5	26.3	15.4	8.7	30.7	99.8
B.		90.0	90.0	55.0	40.0	35.0	40.0	
C.		1.1	15.7	14.3	6.0	3.1	12.4	52.6

Percentage of galena in concentrates: 52.6.
Ratio of concentration based on original feed: 2.63.
Remarks.—The bedding did not pulsate. The upper third was filled with particles of galena and decreasing numbers below. The ore-column was somewhat mobile in spots, but pulsated quite regularly, and on top of the bedding was a zone which exhibited considerable activity.

TEST 33.—*Galena* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, $\frac{1}{4}$ in. (6.35 mm.).
Kind: Circular-arc cam.
Pulsion: $\frac{1}{4}$ (67.7 mm.) = 2.66 in. per sec.
Suction: $\frac{3}{4}$ (22.5 mm.) = 0.88 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100. thro'100.		Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	0.4	5.5	27.0	24.0	13.1	6.5	23.4	99.9
B.	100.0	100.0	60.0	40.0	35.0	35.0	35.0	
C.	0.4	5.5	16.2	9.6	4.5	2.3	8.0	46.5

Percentage of galena in concentrates: 46.5.
Ratio of concentration based on original feed: 2.32.
Remarks.—The upper half and often more of the bedding pulsated. In this part, also, were many particles of galena. The ore-column was mobile, with the large quartz-grains arranged near the top of the bedding and the smaller sizes above.

TEST 34.—*Galena* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, $\frac{1}{4}$ in. (6.35 mm.).
Kind: Circular-arc cam.
Pulsion: $\frac{3}{4}$ (22.5 mm.) = 0.88 in. per sec.
Suction: $\frac{1}{4}$ (67.7 mm.) = 2.66 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100. thro'100.		Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.		3.0	27.7	19.2	12.3	7.2	30.5	99.9
B.		100.0	85.0	70.0	50.0	50.0	45.0	
C.		3.0	23.8	13.3	6.0	3.6	13.5	63.2

Percentage of galena in concentrates: 63.2.
Ratio of concentration based on original feed: 3.16.
Remarks.—The bedding exhibited very little tendency to pulsate, nor was there

any movement among the grains themselves. The upper half of the bedding was well filled with grains of galena. The particles of ore above the bedding circulated in two opposite orbits, passing down at the front and back end of jig and joining in the center.

TEST 35.—*Galena* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, ¼ in. (6.35 mm.).
Kind: Eccentric.
Pulsion and Suction: ½ (33.9 mm.) = 1.33 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.		0.7	17.5	19.2	18.2	8.7	35.2	99.5
B.		95.0	95.0	70.0	45.0	40.0	50.0	
C.		0.7	16.6	13.3	8.1	3.5	17.5	59.7

Percentage of galena in concentrates: 59.7.
Ratio of concentration based on original feed: 3.
Remarks.—The bedding pulsated slightly, and the upper half was well filled with galena, with decreasing quantities below. The ore-column pulsated regularly, with the largest grains of quartz resting on top of the bedding, decreasing in size above.

TEST 36.—*Galena* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 320. Length, ¼ in. (6.35 mm.).
Kind: Eccentric.
Pulsion and Suction: ½ (67. 7 mm.) = 2.66 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	0.6	7.2	30.1	22.8	12.3	5.6	21.2	99.8
B.	100.0	100.0	65.0	45.0	40.0	40.0	50.0	
C.	0.6	7.2	19.5	10.2	4.9	2.2	10.6	55.2

Percentage of galena in concentrates: 55.2.
Ratio of concentration based on original feed: 2.76
Remarks.—The entire bed pulsated regularly. The upper part of bedding contained many particles of galena.

TEST 37.—*Galena* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 320. Length, ⅛ in. (3.17 mm.).
Kind: Eccentric.
Pulsion and Suction: ½ (33.9 mm.) = 1.33 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.		10.3	27.2	23.1	13.7	7.1	19.0	100.4
B.		100.0	85.0	50.0	40.0	40 0	40.0	
C.		10.3	23.1	11.5	5.5	2.8	7.6	60.8

Percentage of galena in concentrates: 60.8.
Ratio of concentration based on original feed: 3.04.
Remarks.—The upper third of bedding was quite mobile, and filled with particles of galena, decreasing below. The ore-column seemed quite compact, but pulsated regularly.

TEST 38.—*Galena* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 320. Length, ½ in. (12.7 mm.).
Kind: Eccentric.
Pulsion and Suction: ½ (135.5 mm.) = 5.33 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	4.3	16.4	34.6	18.2	9.0	4.6	13.2	100.3
B.	100.0	95.0	50.0	40.0	35.0	40.0	45.0	
C.	4.3	15.6	17.3	7.3	3.1	1.8	5.9	55.3

Percentage of galena in concentrates: 55.3.
Ratio of concentration: 2.76.
Remarks.—Both ore and bedding pulsated regularly, but violently. Considerable of the finest size of galena could be seen in the tailings. The jig worked very rapidly.

TEST 41.—*Sphalerite* 10, *Quartz* 90 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, 1 in. (25.4 mm.).
Kind: Circular-arc cam.
Pulsion: ¼ (270.7 mm.) = 10.66 in. per sec.
Suction: ¾ (90.2 mm.) = 3.55 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	11.8	31.8	30.2	11.0	5.0	2.2	5.2	97.2
B.	20.0	20.0	25.0	30.0	35.0	40.0	50.0	
C.	2.4	6.4	7.5	3.3	1.7	0.8	2.6	24.7

Percentage of sphalerite in concentrates: 24.7.
Ratio of concentration based on original feed: 2.47.
Remarks.—The bedding pulsated very violently, and after the jig was stopped it was found that the surface of the ore-column was 1.5 in. below the tailings-dam.

TEST 42.—*Sphalerite* 10, *Quartz* 90 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, 1 in. (25.4 mm.).
Kind: Circular-arc cam.
Pulsion: ¾ (90.2 mm.) = 3.55 in. per sec.
Suction: ¼ (270.7 mm.) = 10.66 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	3.6	14.1	36.5	19.2	9.5	4.5	12.2	99.6
B.	60.0	65.0	35.0	30.0	35.0	30.0	40.0	
C.	2.1	9.1	12.7	5.7	3.3	1.4	5.0	39.3

Percentage of sphalerite in concentrates: 39.3.
Ratio of concentration based on original feed: 3.93.
Remarks.—The bedding-grains were carried up from bottom to top, circulating in that way as by convection-currents. The ore-column was in active agitation, and the bedding and the ore were not separated by a clearly defined and horizontal line.

Test 43.—*Sphalerite* 10, *Quartz* 90 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, 1 in. (25.4 mm.).
Kind: Involute cam.
Pulsion: $\frac{1}{3}$ (203.2 mm.) = 8 in. per sec.
Suction: $\frac{2}{3}$ (101.6 mm.) = 4 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100. thro' 100.		Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	7.6	26.5	34.5	13.7	6.7	3.1	8.1	100.2
B.	25.0	35.0	25.0	25.0	35.0	30.0	40.0	
C.	1.9	9.2	8.6	3.4	2.2	0.9	3.2	29.4

Percentage of sphalerite in concentrates: 29.4.
Ratio of concentration based on original feed: 2.94.
Remarks.—The jig-bed pulsated violently. The bedding did not tend to move in convection-currents, as in Test 42. Between the bedding and the ore-column was a very active zone 0.5 in wide, in which the mineral particles moved in all directions and with great rapidity.

Test 44.—*Sphalerite* 10, *Quartz* 90 *per cent.*

Stroke: Cam-shaft rev. per min, 160. Length, 1 in. (25.4 mm.).
Kind: Involute cam.
Pulsion: $\frac{2}{3}$ (101.6 mm.) = 4 in. per sec.
Suction: $\frac{1}{3}$ (203.2 mm.) = 8 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100. thro' 100.		Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	3.4	17.5	40.1	17.1	8.7	4.0	9.2	100.0
B.	85.0	60.0	30.0	25.0	35.0	30.0	40.0	
C.	3.0	10.5	12.0	4.2	2.9	1.2	3.8	37.6

Percentage of sphalerite in concentrates: 37.6
Ratio of concentration based on original feed: 3.76.
Remarks.—The movement of the jig-bed was very similar to Test 42. The bedding-grains were not only carried from the bottom of the bedding-column itself, but many rose to the top of the ore-column, and a few of the lightest were carried off with the tailings. The grains of quartz could be seen very plainly rolling down with the larger bedding-grains and being carried into the hutch.

Test 45.—*Sphalerite* 10, *Quartz* 90 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, 1 in. (25.4 mm.).
Kind: Eccentric.
Pulsion and Suction: $\frac{1}{2}$ (135.5 mm.) = 5.33 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100. thro' 100.		Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	5.3	17.0	32.2	18.0	9.2	4.7	13.3	99.7
B.	85.0	80.0	45.0	40.0	35.0	40.0	45.0	
C.	4.6	13.6	14.4	7.2	3.3	1.8	5.8	50.7

Percentage of sphalerite in concentrates: 50.7.
Ratio of concentration based on original feed: 5.07.
Remarks.—Both the bedding and the ore-column pulsated regularly. Each formed distinct and well-defined layers. The jig worked very rapidly.

Test 46.—*Sphalerite* 10, *Quartz* 90 *per cent.*

Stroke : Cam-shaft rev. per min., 160. Length, ½ in. (12.7 mm.).
Kind : Eccentric.
Pulsion and Suction ½ (67.7 mm.) = 2.66 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	3.4	15.7	29.7	19.0	11.4	5.8	15.0	100.0
B.	100.0	90.0	50.0	35.0	30.0	30.0	40.0	
C.	3.4	14.4	15.0	6.6	3.3	1.8	6.0	50.5

Percentage of sphalerite in concentrates : 50.5.
Ratio of concentration based on original feed : 5.05.
Remarks.—The upper half to three-fourths of the bedding pulsated regularly, the bottom grains were almost stationary. The lower part of the ore-column consisted of the largest particles of quartz, with smaller and smaller grains to the top.

Test 47.—*Sphalerite* 10, *Quartz* 90 *per cent.*

Stroke : Cam-shaft rev. per min., 160. Length, ½ in. (12.7 mm.).
Kind : Circular-arc cam.
Pulsion : ¼ (135.5 mm.) = 5.33 in. per sec.
Suction : ¾ (45.2 mm.) = 1.77 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	7.0	19.8	34.0	18.8	8.5	3.7	8.0	99.8
B.	60.0	55.0	30.0	25.0	30.0	30.0	40.0	
C.	4.2	11.0	10.2	4.7	2.5	1.2	3.2	37.0

Percentage of sphalerite in concentrates : 37.0.
Ratio of concentration based on original feed : 3.70.
Remarks.—The entire bed moved *en masse*, the top of the column having a longer amplitude of vibration and requiring a longer time for its completion than the grains nearer the bottom. The jig worked rapidly.

Test 48.—*Sphalerite* 10, *Quartz* 90 *per cent.*

Stroke : Cam-shaft rev. per min., 160. Length, ½ in. (12.7 mm.).
Kind : Circular-arc cam.
Pulsion : ¾ (45.2 mm.) = 1.77 in. per sec.
Suction : ¼ (135.5 mm.) = 5.33 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	1.1	7.0	35.0	24.6	12.2	5.1	15.0	100.0
B.	90.0	75.0	30.0	25.0	30.0	30.0	35.0	
C.	1.0	5.2	10.5	6.2	3.6	1.5	5.2	33.2

Percentage of sphalerite in concentrates : 33.2.
Ratio of concentration based on original feed : 3.32.
Remarks.—The bedding pulsated, but not regularly, and tended to thicken in the middle and thin down at the ends. The grains at the bottom of the ore-column were in very active agitation, but it was found that these grains were really describing two distinct orbits.

TEST 49.—*Sphalerite* 10, *Quartz* 90 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, $\frac{1}{2}$ in. (12.7 mm.).
Kind: Involute cam.
Pulsion: $\frac{4}{5}$ (101.6 mm.) = 4 in. per sec.
Suction: $\frac{2}{5}$ (50.8 mm.) = 2 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	9.6	20.0	31.0	18.5	8.5	3.7	7.3	98.6
B.	50.0	65.0	35.0	25.0	30.0	30.0	40.0	
C.	4.8	13.0	10.8	4.2	2.5	1.1	2.8	39.2

Percentage of sphalerite in concentrates: 39.2.

Ratio of concentration based on original feed: 3.92.

Remarks.—The upper two-thirds of the bedding and the entire ore-column pulsated regularly. As noted before, the grains nearest the top had a longer amplitude of vibration and required a longer time to complete it. The lower one-third of the bedding was quite stationary.

TEST 50.—*Sphalerite* 10, *Quartz* 90 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, $\frac{1}{2}$ in. (12.7 mm.).
Kind: Involute cam.
Pulsion: $\frac{2}{5}$ (50.8 mm.) = 2 in. per sec.
Suction: $\frac{4}{5}$ (101.6 mm.) = 4 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	0.8	8.7	41.4	22.0	10.0	4.8	11.6	99.3
B.	100.0	65.0	25.0	25.0	25.0	35.0	40.0	
C.	0.8	5.8	10.2	5.5	2.5	1.6	4.6	31.0

Percentage of sphalerite in concentrates: 31.0.

Ratio of concentration based on original feed: 3.1.

Remarks.—The bedding pulsated slightly, and the grains shifted positions as in convection-currents. A zone between the bedding and the ore-column moved much as noted in Test 40. The ore-column above this zone pulsated regularly, although the ore-column was not very mobile.

TEST 51.—*Sphalerite* 10, *Quartz* 90 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, $\frac{1}{4}$ in. (6.35 mm.).
Kind: Circular-arc cam.
Pulsion: $\frac{1}{4}$ (67.7 mm.) = 2.66 in. per sec.
Suction: $\frac{3}{4}$ (22.5 mm.) = 0.88 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	1.4	9.5	44.5	22.2	9.4	4.6	8.0	99.6
B.	90.0	80.0	25.0	25.0	35.0	30.0	35.0	
C.	1.2	7.6	11.0	5.5	3.2	1.5	2.8	32.8

Percentage of sphalerite in concentrates: 32.8.

Ratio of concentration based on original feed: 3.28.

Remarks.—The bedding and with it the ore-column pulsated *en masse*. Taking the entire column of bedding and ore as a whole, the top had a much longer amplitude of vibration, and required a longer time in which to complete it.

Test 52.—*Sphalerite* 10, *Quartz* 90 *per cent.*

Stroke : Cam-shaft rev. per min., 160. Length, $\frac{1}{4}$ in. (6.35 mm.).
Kind : Circular-arc cam.
Pulsion : $\frac{3}{4}$ (22.5 mm.) = 0.88 in. per sec.
Suction : $\frac{1}{4}$ (67.7 mm.) = 2.66 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100.	thro' 100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	2.0	11.6	31.8	22.7	11.6	5.5	14.6	99.8
B.	100.0	95.0	45.0	35.0	35.0	35.0	35.0	
C.	2.0	11.0	14.4	8.0	4.2	1.6	4.9	46.1

Percentage of sphalerite in concentrates : 46.1.
Ratio of concentration based on original feed : 4.61.
Remarks.—Movement of jig-bed very similar to that of Test 48, but to a less extent.

Test 53.—*Sphalerite* 10, *Quartz* 90 *per cent.*

Stroke : Cam-shaft rev. per min., 160. Length, $\frac{1}{4}$ in. (6.35 mm.).
Kind : Involute cam.
Pulsion : $\frac{1}{3}$ (50.8 mm.) = 2 in. per sec.
Suction : $\frac{2}{3}$ (25.4 mm.) = 1 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100.	thro' 100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	0.6	9.8	30.0	23.8	13.0	7.5	14.8	99.5
B.	100.0	90.0	55.0	40.0	35.0	30.0	35.0	
C.	0.6	9.0	16.5	9.5	4.5	2.1	5.2	47.4

Percentage of sphalerite in concentrates : 47.4.
Ratio of concentration based on original feed : 4.74.
Remarks.—The bedding pulsated in the upper third and half, and was quite mobile as well. The lower part, however, was stationary. The line between bedding and ore was horizontal and straight. The ore-column pulsated regularly—the top for a greater distance, and for a longer time, as before.

Test 54.—*Sphalerite* 10, *Quartz* 90 *per cent.*

Stroke : Cam-shaft rev. per min., 160. Length, $\frac{1}{4}$ in. (6.35 mm.).
Kind : Involute cam.
Pulsion : $\frac{2}{3}$ (25.4 mm.) = 1 in. per sec.
Suction : $\frac{1}{3}$ (50.8 mm.) = 2 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100.	thro' 100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	0.8	5.4	31.2	28.6	13.5	6.7	14.7	100.9
B.	100.0	90.0	50.0	30.0	35.0	30.0	35.0	
C.	0.8	4.8	15.5	8.6	4.5	2.1	4.9	41.2

Percentage of sphalerite in concentrates : 41.2.
Ratio of concentration based on original feed : 4.12.
Remarks.—The entire bedding was practically stationary, did not pulsate, nor was it mobile. The interstitial spaces in the upper part of bedding filled with grains of sphalerite. The ore-column pulsated *en masse* and was fairly mobile.

TEST 55.—*Sphalerite* 10, *Quartz* 90 *per cent.*

Stroke : Cam-shaft rev. per min., 160. Length, ¼ in. (6.35 mm.).
Kind : Eccentric.
Pulsion and Suction : ½ (33.9 mm.) = 1.33 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100.	thro' 100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	0.4	7.7	29.2	19.6	15.0	8.2	19.5	99.6
B.	90.0	95.0	70.0	45.0	40.0	35.0	35.0	
C.	0.4	7.3	20.6	9.0	6.0	2.8	6.6	52.7

Percentage of sphalerite in concentrates : 52.7.
Ratio of concentration based on original feed : 5.27.
Remarks.—The upper third of the bedding was mobile, but the lower two-thirds was quite fixed. The ore-column pulsated regularly, together with the upper third of bedding. The line between the ore-column and the bedding was clearly marked.

TEST 61.—*Sphalerite* 20, *Quartz* 80 *per cent.*

Stroke : Cam-shaft rev. per min., 160. Length, 1 in. (25.4 mm.).
Kind : Circular-arc cam.
Pulsion : ¼ (270.7 mm.) = 10.66 in. per sec.
Suction : ¾ (90.2 mm.) = 3.55 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100.	thro' 100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	6.3	28.2	33.6	12.6	6.0	2.7	9.8	99.2
B.	45.0	35.0	30.0	30.0	35.0	40.0	50.0	
C.	2.7	9.8	10.0	3.8	2.1	1.0	4.5	33.9

Percentage of sphalerite in concentrates : 33.9.
Ratio of concentration based on original feed : 1.7.
Remarks.—The entire jig-bed pulsated very violently. The feed was very fast, a large amount of hutch-work was made, and the tailings contained considerable fine mineral. The fine quartz could be seen sifting through the bedding.

TEST 62.—*Sphalerite* 20, *Quartz* 80 *per cent.*

Stroke : Cam-shaft rev. per min., 160. Length, 1 in. (25.4 mm.).
Kind : Circular-arc cam.
Pulsion : ¾ (90.2 mm.) = 3.55 in. per sec.
Suction : ¼ (270.7 mm.) = 10.66 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100.	thro' 100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	1.9	13.2	38.0	18.3	9.7	4.6	14.0	99.7
B.	90.0	75.0	35.0	40.0	40.0	50.0	50.0	
C.	1.8	9.7	13.3	7.4	3.8	2.3	7.0	45.3

Percentage of sphalerite in concentrates : 45.2.
Ratio of concentration based on original feed : 2.26.
Remarks.—Behavior of jig-bed similar to Test 41.

TEST 63.—*Sphalerite* 20, *Quartz* 80 *per cent.*

Stroke : Cam-shaft rev. per min., 160. Length, 1 in. (25.4 mm.).
Kind : Involute cam.
Pulsion : $\frac{1}{3}$ (203.2 mm.) = 8 in. per sec.
Suction : $\frac{2}{3}$ (101.6 mm.) = 4 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	4.8	21.5	36.8	15.1	7.8	3.3	10.2	99.5
B.	70.0	50.0	35.0	40.0	45.0	50.0	60.0	
C.	3.3	10.7	12.9	6.0	3.4	1.6	6.0	43.9

Percentage of sphalerite in concentrates : 43.9.
Ratio of concentration based on original feed : 2.20.
Remarks.—Behavior of jig-bed similar to Test 43.

TEST 64.—*Sphalerite* 20, *Quartz* 80 *per cent.*

Stroke : Cam-shaft rev. per min., 160. Length, 1 in. (25.4 mm.).
Kind : Involute cam.
Pulsion : $\frac{2}{3}$ (101.6 mm.) = 4 in. per sec.
Suction : $\frac{1}{3}$ (203.2 mm.) = 8 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	2.3	17.2	37.8	17.3	8.5	4.1	12.8	100.0
B.	85.0	45.0	35.0	40.0	40.0	50.0	55.0	
C.	1.9	7.6	13.1	6.8	3.4	2.0	6.8	41.6

Percentage of sphalerite in concentrates : 41.6.
Ratio of concentration based on original feed : 2.08.
Remarks.—Movement of jig-bed similar to Test 44.

TEST 65.—*Sphalerite* 20, *Quartz* 80 *per cent.*

Stroke : Cam-shaft rev. per min., 160. Length, 1 in. (25.4 mm.).
Kind : Eccentric.
Pulsion and Suction : $\frac{1}{2}$ (135.5 mm.) = 5.33 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	2.5	14.2	34.6	18.6	10.0	4.5	15.0	99.4
B.	90.0	80.0	50.0	40.0	45.0	50.0	60.0	
C.	2.2	11.2	17.2	7.4	4.5	2.2	9.0	53.7

Percentage of sphalerite in concentrates : 53.7.
Ratio of concentration based on original feed : 2.7.
Remarks.—Movement of jig-bed similar to Test 45.

TEST 66.—*Sphalerite* 20, *Quartz* 80 *per cent.*

Stroke : Cam-shaft rev. per min., 160. Length, $\frac{1}{2}$ in. (12.7 mm.).
Kind : Eccentric.
Pulsion and Suction : $\frac{1}{2}$ (67.7 mm.) = 2.66 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	2.5	12.4	33.0	19.0	11.0	4.9	17.2	100.0
B.	90.0	85.0	60.0	45.0	45.0	50.0	50.0	
C.	2.2	10.5	20.0	8.5	5.0	2.5	8.6	57.3

Percentage of sphalerite in concentrates : 57.3.
Ratio of concentration based on original feed : 2.8.
Remarks.—Movement of jig-bed similar to Test 46.

TEST 67.—*Sphalerite* 20, *Quartz* 80 *per cent.*

Stroke : Cam-shaft rev. per min., 160. Length, ½ in. (12.7 mm.).
Kind : Circular-arc cam.
Pulsion : ¼ (135.5 mm.) = 5.33 in. per sec.
Suction : ¾ (45.2 mm.) = 1.77 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	6.1	16.3	32.4	16.9	8.7	4.0	14.2	98.6
B.	65.0	60.0	45.0	35.0	40.0	45.0	50.0	
C.	3.9	9.6	14.6	6.0	3.4	1.8	7.1	46.4

Percentage of sphalerite in concentrates : 46.4.
Ratio of concentration based on original feed : 2.32.
Remarks.—The bedding and the ore-column pulsated, and the bottom grains of bedding much more than in Test 66.

TEST 68.—*Sphalerite* 20, *Quartz* 80 *per cent.*

Stroke : Cam-shaft rev. per min., 160. Length, ½ in. (12.7 mm.).
Kind : Circular-arc cam.
Pulsion : ¾ (45.2 mm.) = 1.77 in. per sec.
Suction : ¼ (135.5 mm.) = 5.33 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	1.2	8.1	36.4	21.6	11.2	5.6	16.5	100.6
B.	95.0	80.0	50.0	40.0	40.0	45.0	50.0	
C.	1.1	6.4	18.2	8.6	4.4	2.5	8.2	49.4

Percentage of sphalerite in concentrates : 49.4.
Ratio of concentration based on original feed : 2.47.
Remarks.—Movement of jig-bed similar to Test 48.

TEST 69.—*Sphalerite* 20, *Quartz* 80 *per cent.*

Stroke : Cam-shaft rev. per min., 160. Length, ½ in. (12.7 mm.).
Kind : Involute cam.
Pulsion : ⅓ (101.6 mm.) = 4 in. per sec.
Suction : ⅔ (50.8 mm.) = 2 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	5.2	18.2	32.7	17.7	9.2	4.1	12.5	99.6
B.	70.0	75.0	45.0	45.0	50.0	50.0	55.0	
C.	3.5	13.5	14.8	7.9	4.6	2.0	6.8	53.1

Percentage of sphalerite in concentrates : 53.1.
Ratio of concentration based on original feed : 2.65.
Remarks.—The entire jig-bed moved *en masse*, and was very mobile. As in all cases of this kind, the upper part of the ore-column had a longer amplitude of vibration and required a longer time in which to complete it than the grains (whether bedding or ore) nearer the bottom.

TEST 70.—*Sphalerite* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, ½ in. (12.7 mm.).
Kind: Involute cam.
Pulsion: ⅜ (50.8 mm.) = 2 in. per sec.
Suction: ⅛ (101.6 mm.) = 4 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	1.1	8.7	40.6	20.8	10.5	4.5	13.7	99.9
B.	95.0	80.0	45.0	35.0	40.0	50.0	60.0	
C.	1.0	6.8	16.2	7.4	4.2	2.2	8.1	45.9

Percentage of sphalerite in concentrates: 45.9.
Ratio of concentration based on original feed: 2.29.
Remarks.—Movement of jig-bed similar to Test 50.

TEST 71.—*Sphalerite* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, ¼ in. (6.35 mm.).
Kind: Circular-arc cam.
Pulsion: ¼ (67.7 mm.) = 2.66 in. per sec.
Suction: ¾ (22.5 mm.) = 0.88 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	1.1	12.0	38.5	22.0	10.0	4.6	11.2	99.4
B.	95.0	80.0	40.0	35.0	45.0	50.0	55.0	
C.	1.0	9.6	14.9	7.7	4.7	2.3	6.0	46.2

Percentage of sphalerite in concentrates: 46.2.
Ratio of concentration based on original feed: 2.31.
Remarks.—Movement of jig-bed similar to Test 51.

TEST 72.—*Sphalerite* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, ¼ in. (6.35 mm.).
Kind: Circular-arc cam.
Pulsion: ¾ (22.5 mm.) = 0.88 in. per sec.
Suction: ¼ (67.7 mm.) = 2.66 in. per sec.

On mesh....	12.	20.	40.	60.	80.	100.	thro'100.	Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	1.8	10.6	31.2	22.2	12.4	6.4	15.2	99.8
B.	100.0	95.0	70.0	50.0	45.0	50.0	60.0	
C.	1.8	9.9	21.7	11.1	5.6	3.8	9.0	62.9

Percentage of sphalerite in concentrates: 62.9.
Ratio of concentration based on original feed: 3.14.
Remarks.—The entire mass except the lower part of the bedding pulsated *en masse*, and the ore-column seemed quite mobile.

TEST 73.—*Sphalerite* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, $\frac{1}{4}$ in. (6.35 mm.).
Kind: Involute cam.
Pulsion: $\frac{1}{3}$ (50.8 mm.) = 2 in. per sec.
Suction: $\frac{2}{3}$ (25.4 mm.) = 1 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100. thro'100.		Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	1.3	8.7	27.2	20.6	14.1	7.1	20.8	99.8
B.	100.0	100.0	80.0	50.0	50.0	55.0	60.0	
C.	1.3	8.7	21.6	10.3	7.0	3.8	12.6	65.3

Percentage of sphalerite in concentrates: 65.3.
Ratio of concentration based on original feed: 3.26.
Remarks.—The upper two-thirds of bedding and the entire ore-column pulsated. Ore-column mobile.

TEST 74.—*Sphalerite* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, $\frac{1}{4}$ in. (6.35 mm.).
Kind: Involute cam.
Pulsion: $\frac{2}{3}$ (25.4 mm.) = 1 in. per sec.
Suction: $\frac{1}{3}$ (50.8 mm.) = 2 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100. thro'100.		Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	0.9	6.5	24.5	20.6	15.5	7.9	24.1	100.0
B.	100.0	100.0	85.0	65.0	55.0	60.0	60.0	
C.	0.9	6.5	20.9	13.3	8.5	4.8	14.4	69.3

Percentage of sphalerite in concentrates: 69.3.
Ratio of concentration based on original feed: 3.46.
Remarks.—The upper third of bedding together with the ore-bed pulsated *en masse*. The lower two-thirds of bedding was quite fixed in position. The top of ore-column, as before, had a longer amplitude.

TEST 75.—*Sphalerite* 20, *Quartz* 80 *per cent.*

Stroke: Cam-shaft rev. per min., 160. Length, $\frac{1}{4}$ in. (6.35 mm.)
Kind: Eccentric.
Pulsion and Suction: $\frac{1}{2}$ (33.9 mm.) = 1.33 in. per sec.

On mesh.....	12.	20.	40.	60.	80.	100. thro'100.		Total.
Size in mm..	1.66	0.97	0.42	0.26	0.21	0.16	0.16	
A.	0.2	4.2	29.1	20.2	13.8	7.1	24.8	99.4
B.	90.0	90.0	75.0	65.0	55.0	60.0	60.0	
C.	0.2	3.8	21.8	13.0	7.4	4.2	15.0	65.4

Percentage of sphalerite in concentrates: 65.4.
Ratio of concentration based on original feed: 3.27.
Remarks.—The upper third of bedding together with the ore-column pulsated *en masse*. The lower two-thirds of bedding scarcely moved. The interstitial spaces of the bedding, as with all experiments with the short stroke, filled with mineral.

5. *Discussion of Results.*

It is evident that in a machine so simple as the jig there are a number of variables, and a series of tests may therefore be

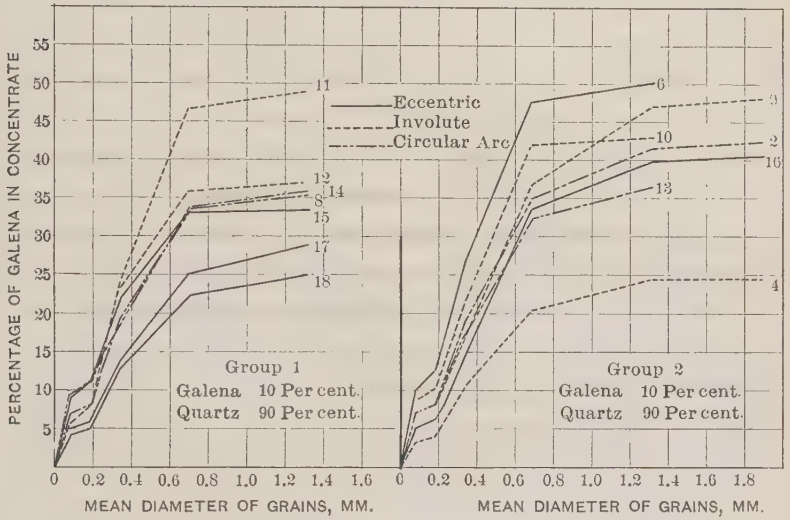


FIG. 6.—CHART OF RESULTS OF GROUPS 1 AND 2 OF CLASS I.

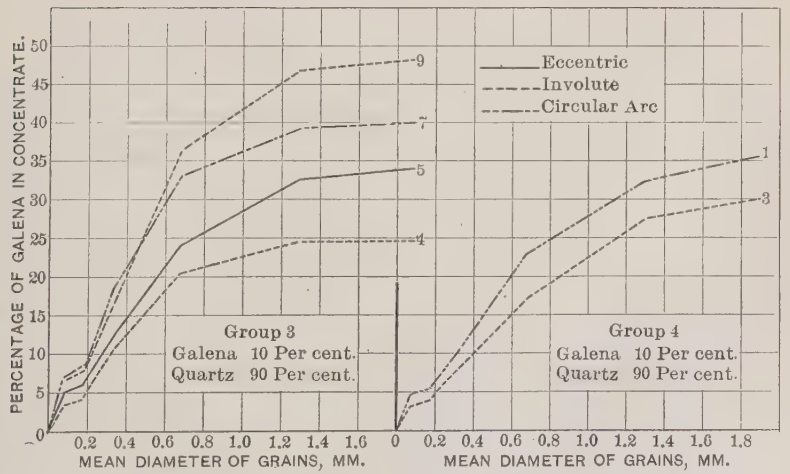


FIG. 7.—CHART OF RESULTS OF GROUPS 3 AND 4 OF CLASS I.

classified according to some one of them. For purposes of discussion, however, the tests that have been conducted are grouped according to the velocity of the rising current of water, or pulsion-currents, as measured by the mean piston-speed. Figs. 6

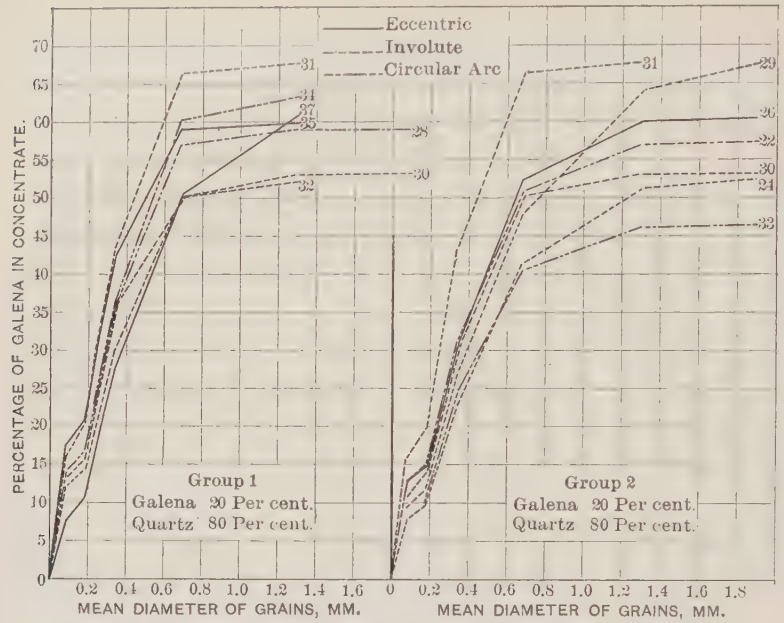


FIG. 8.—CHART OF RESULTS OF GROUPS 1 AND 2 OF CLASS II.

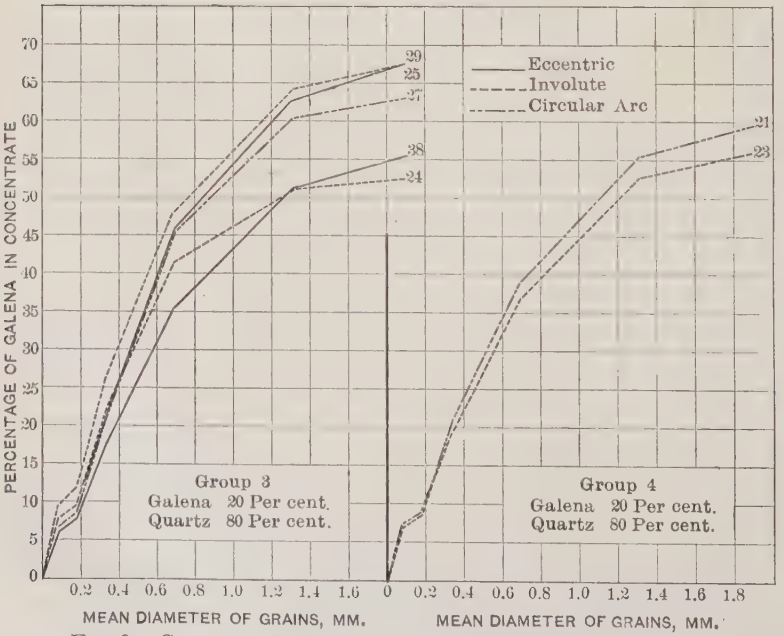


FIG. 9.—CHART OF RESULTS OF GROUPS 3 AND 4 OF CLASS II.

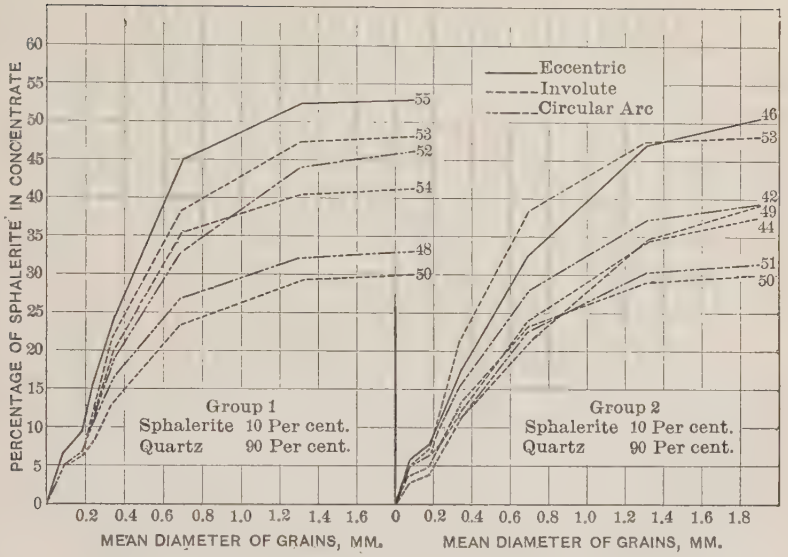


FIG. 10.—CHART OF RESULTS OF GROUPS 1 AND 2 OF CLASS III.

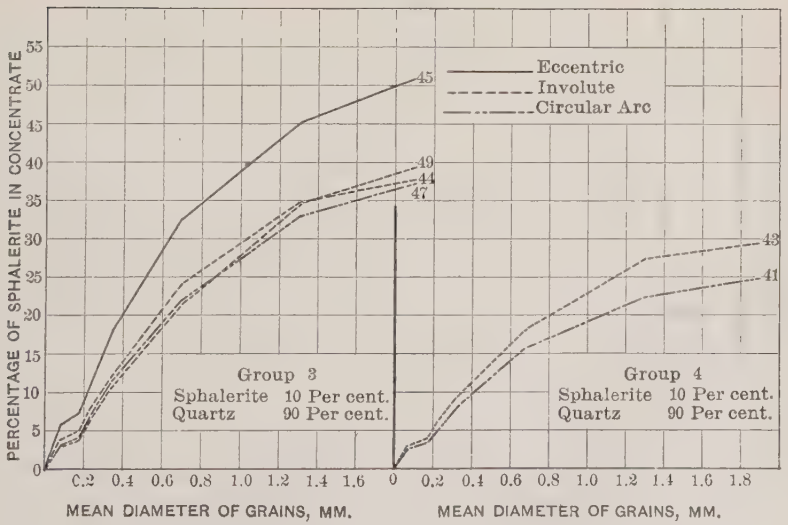


FIG. 11.—CHART OF RESULTS OF GROUPS 3 AND 4 OF CLASS III.

to 13, inclusive, show graphically the results given in row *C*, under each of the experiments, calculated for the mean diameter of the material. Since all material treated on the jig passed through a screen having a square hole, the mean length of the

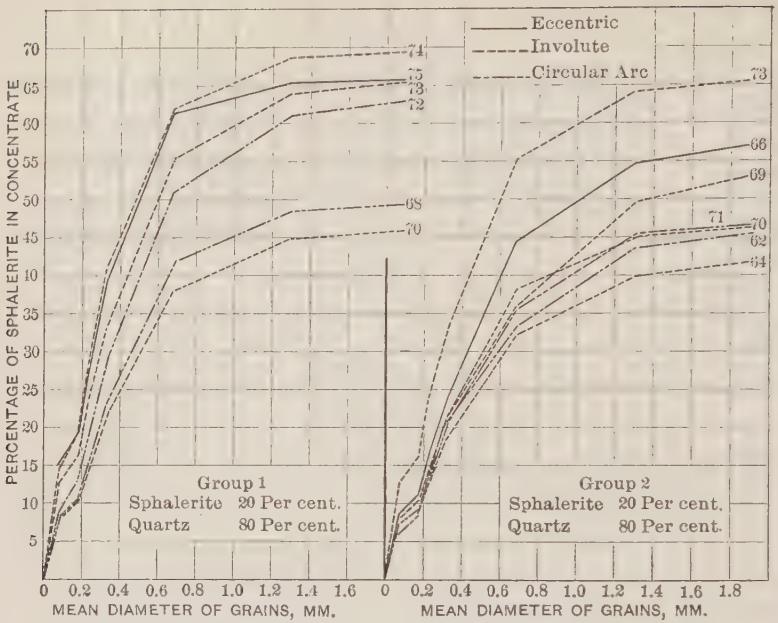


FIG. 12.—CHART OF RESULTS OF GROUPS 1 AND 2 OF CLASS IV.

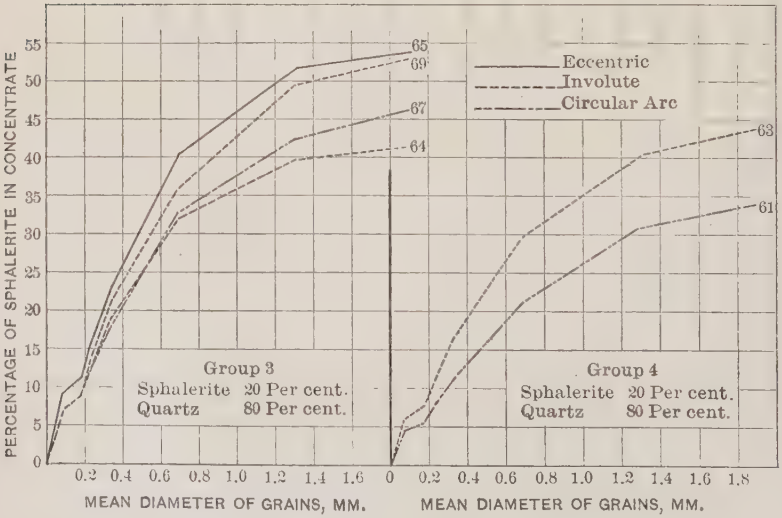


FIG. 13.—CHART OF RESULTS OF GROUPS 3 AND 4 OF CLASS IV.

sides of which was equal to 2.136 mm., and from that as a maximum to the very finest dust, it has been assumed that the

mean diameter of the grain caught on the 1.66-mm. screen is $(2.136 + 1.66) \div 2 = 1.90$ mm.; those passing the 1.66-mm. screen and caught on the 0.97-mm. screen, have a mean diameter of $(1.66 + 0.97) \div 2 = 1.31$ mm., and so on for all the sizes; and, finally, that the material passing the 0.16-mm. screen (the finest used in these tests) had a mean diameter of 0.08 mm. In the curves shown in Figs. 6 to 13 the diameter of the grain has been plotted along the X axis, and the weight of pure mineral (galena or sphalerite) on each screen-size, as given in the record of the tests, laid off on the Y axis. The points thus located have been joined by three classes of lines: the solid lines in all cases represent the results obtained in the tests made with the eccentric cam; the dotted lines, tests with the involute cam; and, finally, the broken lines, tests with the circular-arc cams.

The velocities of the pulsion- or rising-currents, as measured above, have been divided, on purely arbitrary grounds, into four groups: (1) velocities of pulsion 2 in. (50.8 mm.) per sec. and less; (2) velocities of pulsion from 2 to 4 in. (50.8 to 101.6 mm.) per sec.; (3) velocities of pulsion from 4 to 6 in. (101.6 to 152.4 mm.) per sec.; (4) velocities of pulsion from 6 to 10.66 in. (152.4 to 270.7 mm.) per sec. With this arrangement it has so happened that in nearly every case the tests with the involute cam are plotted in two groups. The experiments with the eccentric and circular-arc cams occur once only in each group. Since each pair of minerals has been run with 10 and with 20 per cent. of either galena or blende, two classes are to be distinguished. In all cases under discussion, Classes I. and III. will refer to mixtures containing 10 per cent., and Classes II. and IV. to mixtures with 20 per cent. of the heavy mineral.

In order to facilitate reference, the following classification is given:

Class I. Galena and Quartz. Galena 10, Quartz 90 per cent.

Group 1. Velocities of pulsion from 0 to 2 in. (0 to 50.8 mm.) per sec.

Tests 8, 11, 12, 14, 15, 17, 18.

Group 2. Velocities of pulsion from 2 to 4 in. (50.8 to 101.6 mm.) per sec.

Tests 2, 4, 6, 9, 10, 13, 16.

Group 3. Velocities of pulsion from 4 to 6 in. (101.6 to 152.4 mm.) per sec.

Tests 4, 5, 7, 9.

Group 4. Velocities of pulsion from 6 to 10.66 in. (152.4 to 270.7 mm.) per sec.

Tests 1, 3.

Class II. Galena and Quartz. Galena 20, Quartz 80 per cent.

Group 1. Velocities of pulsion from 0 to 2 in. (0 to 50.8 mm.) per sec.

Tests 28, 30, 31, 32, 34, 35, 37.

Group 2. Velocities of pulsion from 2 to 4 in. (50.8 to 101.6 mm.) per sec.

Tests 22, 24, 26, 29, 30, 31, 33.

Group 3. Velocities of pulsion from 4 to 6 in. (101.6 to 152.4 mm.) per sec.

Tests 24, 25, 27, 29, 38.

Group 4. Velocities of pulsion from 6 to 10.66 in. (152.4 to 270.7 mm.) per sec.

Tests 21, 23.

Class III. Sphalerite and Quartz. Sphalerite 10, Quartz 90 per cent.

Group 1. Velocities of pulsion from 0 to 2 in. (0 to 50.8 mm.) per sec.

Tests 48, 50, 52, 53, 54, 55.

Group 2. Velocities of pulsion from 2 to 4 in. (50.8 to 101.6 mm.) per sec.

Tests 42, 44, 46, 49, 50, 51, 53.

Group 3. Velocities of pulsion from 4 to 6 in. (101.6 to 152.4 mm.) per sec.

Tests 44, 45, 47, 49.

Group 4. Velocities of pulsion from 6 to 10.66 in. (152.4 to 270.7 mm.) per sec.

Tests 41, 43.

Class IV. Sphalerite and Quartz. Sphalerite 20, Quartz 80 per cent.

Group 1. Velocities of pulsion from 0 to 2 in. (0 to 50.8 mm.) per sec.

Tests 68, 70, 72, 73, 74, 75.

Group 2. Velocities of pulsion from 2 to 4 in. (50.8 to 101.6 mm.) per sec.
Tests 62, 64, 66, 69, 70, 71, 73.
Group 3. Velocities of pulsion from 4 to 6 in. (101.6 to 152.4 mm.) per sec.
Tests 64, 65, 67, 69.
Group 4. Velocities of pulsion from 6 to 10.66 in. (152.4 to 270.7 mm.) per sec.
Tests 61, 63.

Class I., Group 1. Galena, 10 per cent. The lowest two ratios of concentration were obtained with two tests with eccentric cam, using a short stroke and a high frequency in Tests 17 and 18; and of these two, Test 18, with only a $\frac{1}{16}$-in. stroke and 400 strokes per min., yields the lowest ratio of the series. An examination of the screen-analysis shows a marked difference between Tests 18 and 15; the longer and slower stroke has caused a larger percentage of the finest size to pass into the hutch; but the shorter and more rapid stroke has increased the percentage of material between 1.31 and 0.69 mm. in the concentrate. In this case, at least, piston-speed does not determine whether the jig-bed will be pulsated, or the proportions of coarse and fine material carried into the hutch.

The highest ratio of concentration is clearly with the involute cam, Test 11, with a pulsion-velocity of 2 in. (50.8 mm.) and suction-velocity of 1 in. (25.4 mm.) per sec. A good catch of fine material is made, and the three largest sizes are of good proportions as to weight and mineral-content; the sudden drop in value for particles of 0.34 mm. diameter should be noted.

Tests 8, 12, 14, and 15 represent the three types of strokes. The weak pulsion and strong suction of Tests 8 and 14 have produced results very similar to those of the eccentric. This style of cam, therefore, between the limits of this group, is not more efficient than the eccentric. In none of the experiments of this group has any material of mean diameter 1.90 mm. been carried through the sieve and into the hutch. It cannot be said that strong suction is superior to moderate suction in saving the fines.

Class I., Group 2. The minimum ratio of concentration is that with the involute cam, Test 4 representing the highest

limit of velocity of pulsion for this group and very strong suction. The strong suction, however, has not resulted in increasing the proportion of fines, probably owing to the fact that there is also a rather high pulsion-velocity. The conditions in this test have been favorable for the recovery of grains of mean diameter of 0.69 mm. (on 0.42-mm. screen). The highest ratio of concentration is obtained in Test 6, an eccentric, with velocity of pulsion and suction 2.66 in. (67.7 mm.) per sec. Test 10, an involute cam, with slow pulsion and rapid suction, gives results similar to Test 6. Test 9 gives very good results, with high pulsion and slow suction, just the reverse of Test 10. In this latter case the strong pulsion is clearly an advantage, resulting in almost as good a saving of fines, and a much larger and cleaner product on the large sizes. Test 2, with circular-arc cam, and Test 16, with eccentric cam, give results quite close. Why Test 4 should differ so materially from Test 2 is not easy to explain. Tests 10 and 4 represent the velocity-limits of the group and show marked differences in results; and in Tests 4 and 9, with the same velocity of pulsion, but different suction-velocity, the strong suction has produced a much smaller percentage of the finest size and contains less galena, although the strong suction has been very effective in drawing material having a mean diameter of 0.69 mm. into the hutch. The eccentric, with short stroke and high frequency, Test 16, gives a low ratio of concentration. In all cases only a small catch is made with sizes larger than about 1 mm., and from 75 to 95 per cent. of the mineral saved in the hutch is of a diameter of 0.69 mm. or less.

In this group the same pulsion-velocity but variable suction-velocity give different results; the high suction-velocity is not of any distinct advantage in increasing the catch of fines or in enriching any of the sizes. The eccentric at proper rotative and pulsion-speeds yields results equal and in most cases superior to an accelerated and retarded stroke.

Class I., Group 3. In this group of four experiments, Test 4, which was also placed with group 2 of this class, and occupied the lowest position, is also the lowest in this group. The highest ratio of concentration is found in Test 9, an involute cam with the same pulsion-velocity as Test 4, but only one-fourth the suction-velocity. As noted under group 2, the

strong suction has resulted in producing a smaller amount of the finest size, and in a decreased percentage of galena in all the sizes. The intermediate positions are marked by an eccentric cam, Test 5, and a circular-arc cam, Test 7, and with the same velocity of pulsion. The strong pulsion and weak suction have resulted in a larger saving of the fine material than in the case of the eccentric, and a somewhat higher ratio of concentration.

It appears, therefore, that in this group the involute cam with strong pulsion and weak suction is the most efficient in producing a high concentrate, and the reverse of these conditions the least efficient; that the circular-arc cam, Test 7, with strong pulsion and weak suction, is somewhat more efficient than the eccentric; and that the same velocities of pulsion yield different results.

It will be noted, also, that all the tests in this group produced some of the coarsest size, and those of strong pulsion and weak suction the largest amount. In comparing this group with group 2 of this series, we find that the maximum ratio has been passed, and that velocities of pulsion more than 4 in. (101.6 mm.) per sec., with the size of material jigged, should not be exceeded.

Class I., Group 4. Only two experiments occur in this group, Tests 1 and 3. Length of stroke in each case 1 in. It appears that the circular-arc cam with the highest velocity of pulsion and least velocity of suction gives a little higher ratio of concentration, but that the results are very much the same. The proper limit for pulsion-velocity has long since been exceeded. Comparing Tests 3 and 4 in the same way, it is found that strong pulsion and weak suction produced practically the same percentages of sieve-sizes, but with strong suction the percentage of heavy material is much reduced. Even with these high velocities of pulsion, a strong suction is not an advantage, in increasing either the amount of fine material drawn into the hutch or the percentage of heavy mineral.

Class II., Group 1. Galena, 20 per cent. Of the seven tests in the group, the involute cam, Test 31, with velocity of pulsion at the maximum limit of the group, gives the highest ratio of concentration. The same was true under Class I. The final minimum ratio is indicated by the reciprocal of Test 31, with

weak pulsion and strong suction. With material up to 0.69 mm. in diameter, the eccentric with short and rapid stroke gives the lowest ratio in Test 37, while at the same piston-speed at twice the length of stroke and half the number of rev. per min., the values are very close to the maximum in Test 35.

The involute cam in Test 30, with weak pulsion and strong suction, produces similar results but at different velocities, but at the same ratio to Test 32. In this case the strongest suction has drawn a larger percentage of the fine stuff into the hutch, but has not enriched it.

The two circular-arc cams, Tests 28 and 34, with weak pulsion and strong suction, give similar results, in which about 30 per cent. of the hutch-product passes through a 100-mesh (0.16 mm.) sieve. But again, in Test 31, with the involute cam, strong pulsion and weak suction, a larger percentage of fine material is drawn into the hutch. The eccentric gives about the same percentage of fines as in Test 31.

It may be said for this group that the eccentric at the proper length and rotative velocity gives excellent results, and is generally superior to an accelerated or retarded stroke. The same pulsion-velocities give different results.

Class II., Group 2. The minimum ratio of concentration is indicated by the circular-arc cam, Test 33, with strong pulsion and weak suction. The maximum is attained with an involute cam, Test 31, with rapid pulsion and weak suction. The involute cam has already been considered under the first group. Test 29, also an involute, under the same conditions, gives good ratios, but the higher pulsion-velocity results in a smaller saving of the very fine material; larger sizes appear more abundantly in the hutch, however. Tests 24 and 30, involute cams with suction in excess of pulsion, give final results that are very close, but the strongest suction, Test 24, yields relatively less fine and more coarse material than the weaker suction.

It will be noted, further, that with the exception of Test 31, some material larger than 1.66 mm. is found in all the products. The circular-arc cam with very high velocity of suction has produced a relatively high percentage of the finest size. The eccentric, Test 26, gives good average results—a large percentage by weight of the finest, and containing at least an average percentage of galena. With the exception of Tests 29

and 31, the five other tests are, in general, much the same. Both of these tests have been classed in other groups. Of the three highest ratios of concentration, two have low suction-velocity and the third has equal pulsion and suction.

Class II., Group 3. Of these five tests the minimum is found in Test 24, repeating the conditions of Test 4. It will be noted, however, that until the size next the largest is reached, the lowest ratio is indicated by the eccentric, Test 38, with short stroke and high rotative speed. Test 29, involute cam, strong pulsion and weak suction, and Test 25, eccentric, at the same piston-speed as Test 38, give almost the same final results.

The circular-arc cam, Test 27, strong pulsion and weak suction, produces results very similar to those of Tests 25 and 29.

In all cases with these high pulsion-velocities, more material having a diameter of 1.31 and 1.90 mm., and correspondingly less of the finer sizes, have been obtained. The advantages of high suction-velocity over those of pulsion are not apparent.

Class II., Group 4. An examination of the two tests in this group indicates at once a close correspondence. The maximum limit for pulsion-velocity has been passed, but it appears that with the richer feed these velocities vary between considerably wider limits than with the poorer material.

In general, it may be said for all the tests, that for each condition under which jigging takes place, certain sizes—20- or 40-mesh (0.97 or 0.42 mm.)—are very rich, and then on smaller sizes a very violent drop in the percentage of galena takes place. This will be noticed for all tests on galena and blende as well. Also, that moderate suction and stronger pulsion give better results than the reverse. The strong pulsion usually results in a larger yield of the coarser sizes of higher percentage in mineral, and the fines are saved almost equally as well. In most cases the eccentric, at the proper length of stroke and rotative speed, is equal and usually superior to accelerated or retarded stroke; but when the stroke becomes too short and the rotative speed high, the ratio falls off. Observations on the behavior of the bed under these conditions showed that the bedding and ore-column pulsated, although at the same piston-speed with the longer stroke no movement in the bedding took place. This indicates that, with very sudden impulses to the ore-column, the water acts more like a solid than a liquid, and that mineral-

particles are not subject to the full force of a rising current of water, but that the material is sifted down through the interstitial spaces of the bedding. Possibly another cause is at work, as noted in the behavior of the bed during the long strokes. Here the top of the bed pulsated for a longer time, and had a longer amplitude of vibration, and therefore the grains on the bottom came to rest sooner than those above, which would tend to limit the size of the particles passing into the hutch; and the longer and slower the stroke above the limits which will move the grains, the more pronounced will this differential motion be, and with it the increased perfection of the classification that must take place.

Class III., Group 1. Sphalerite, 10 per cent. Here the lowest ratio is found in Test 50, with weak pulsion and strong suction. An examination of the weights and percentages of Tables IV. and V. shows, however, that only relatively small amounts of the finest sizes are secured; but material of 0.69 mm. (on 40-mesh) is recovered to an amount equal to about 41 per cent., while material larger or smaller than this size is not materially increased. Test 48, under similar conditions, gives similar results. Both of these tests indicate that strong suction, within the limits of this group, is not advantageous. Test 54, with the same ratio of pulsion and suction, but only one-half the intensity, gives a higher ratio of concentration and a slightly better recovery in the finest sizes. Test 52, under analogous conditions, gives somewhat similar results, except material on 20-mesh (1.31 mm. mean diameter). Test 53, involute cam, with strong pulsion and weak suction, gives very good results. Test 55, eccentric, gives the best results of all. In this case, not only a high percentage of the finest sizes of fair mineral-content was obtained, but the coarse sizes also were well represented, containing a high percentage of sphalerite, which accounts chiefly for the high ratio of concentration.

It may be said for this group that the eccentric easily yields the best results; that strong suction and weak pulsion give the lowest, and weak suction and strong pulsion an improved ratio of concentration.

Class III., Group 2. An inspection of the tests in this group shows that the lowest ratio of concentration is indicated by Test 50, an involute cam, with strong suction and weak pul-

sion, already considered in Class III., Group 1; and very near it is Test 51, a circular-arc cam, with strong pulsion and weak suction, resulting in the production of very small amounts of the finest sizes; but nearly 45 per cent. of material on the 40-mesh (0.69 mm. mean diameter). Tests 42, 44, and 49 give results in the final ratios that are close together, but differing in the details. Test 42, circular-arc cam, with moderate pulsion and strong suction, and Test 49, involute cam, with strong pulsion and weak suction, give practically the same final result; and Test 44, involute cam, with moderate pulsion and strong suction, similar results.

The two higher ratios are those of Test 46, eccentric, and Test 53, involute cam, with strong pulsion, but the lowest for the group, and less suction. An inspection of the records of the experiments shows that, with the relatively low pulsion-velocity used, these two tests yielded relatively less of the coarsest sizes, but increased amounts of the finest sizes.

The superiority of the eccentric over the other forms of stroke is at once evident. In the case of all styles of stroke, the same pulsion-velocity gives final results much the same.

Class III., Group 3. Of the four tests grouped here, three are almost identical—namely, Tests 44, 47, and 49; and of these three, two have already been considered in Class III., Group 2. Test 47, circular-arc cam, with strong pulsion and weak suction, and Test 49, also strong pulsion and weak suction, produce about the same results as very strong suction and weaker pulsion, but in which, however, the pulsion-velocity is about the same. This indicates that the velocity of pulsion is the principal determining factor.

Class III., Group 4. The two tests in this group are very closely related. It is evident that the proper velocity of pulsion has been passed. The records of the experiments show that, at these high velocities, the coarse sizes readily pass into the hutch, but at the same time the percentage of mineral is much decreased, and much of the fine material is lost.

An examination of the four groups indicates that in the fourth the maximum velocity of pulsion for good work has been exceeded, but in the other three groups the best velocity is not so clearly indicated. With the three eccentrics good ratios have been secured in each of the groups, and this is

also the most efficient of the three types of stroke. A high pulsion-velocity is very efficient in saving material that rests on 20- and 12-mesh (1.31 and 1.90 mm. mean diameter of grain), but, on sizes smaller than these, less so than decreased velocities. A high suction-velocity is not generally more efficient in recovering the finest sizes than a more moderate one.

Class IV., Group 1. Sphalerite, 20 per cent. A comparison with the corresponding group of Class II. shows many features in common. The lowest ratio of concentration is found in Test 70, and next to it Test 68, both weak pulsion and strong suction. Test 72, the reciprocal of Test 68, gives better results. Tests 73 and 74, reciprocals of each other, indicate that between these velocities the involute cam is very efficient. Test 74, with strong suction, gives the highest ratio of the group.

Class IV., Group 2. Some differences as compared with the corresponding group of Class I. are found here. The minimum ratio of concentration is marked by Test 64, involute cam, with moderate pulsion and strong suction. Tests 62, 70 and 71—62 and 70, circular-arc cams and involute, respectively, with weak pulsion and strong suction, and Test 71, with strong pulsion and weak suction—give results that do not differ materially, indicating once more that even though the suction-velocity differs widely, the final results will not differ widely if the pulsion-velocities are close together. The eccentric, Test 66, shows a good ratio. Test 73, an involute cam, with stronger pulsion than suction, gives the maximum ratio for the group.

Class IV., Group 3. The four tests in this group give results agreeing very closely with the corresponding group of Class I., and the observations made under that group apply here.

Class IV., Group 4. A glance shows at once that this group agrees exactly with the corresponding group under Class I.

An examination of the four groups of Class II. indicates that in the first, with a pulsion-velocity not exceeding 2 in. (50.8 mm.) per sec., the highest ratios are obtained, and that at these velocities by far the largest percentage of the mineral recovered has a mean diameter of 0.69 mm. (through 20-mesh.). As the velocity is increased, more of the coarse sizes appear and less fine material.

For both Classes III. and IV., with sphalerite and quartz, it appears that generally a stronger pulsion-velocity than suction is more efficient in producing a better concentrate, and effects an equally good saving of the fines. The eccentric, between wide velocity-limits, is an efficient type of stroke. Of the two types of cams, the involute is generally the best. An examination of the tests will show that certain types and velocities of strokes are especially suited to the recovery of particles of fixed diameters.

V. Discussion of Pulsion and Suction.

Since the pulsion- and suction-velocity, as measured by the piston-speed, have been the chief variables in this investigation, the question naturally arises: can the exact rôle of each be definitely defined?

The accepted meaning of the terms "pulsion" and "suction" is doubtless familiar to all. A pulsion-current is one acting opposite to gravity, and tending to raise the grain off the jig-sieve; and a suction-current is one acting in the direction of gravity and supplementing it. In both cases, therefore, are reactions caused by the movement of a column of water or other liquid relative to some solid.

When the results of a series of tests are arranged according to the pulsion-velocity in the free part of the jig-column, or, in other words, the piston-speed, even though the suction-velocity differed widely, the final results are quite close together, indicating that the reactions occurring during this cycle determine the final result. With a perfect-fitting piston, given the areas of piston and jig-sieve, length and number of strokes per unit of time, the mean pulsion-velocity in the free or unoccupied section of the jig-column may be accurately determined; and similarly for the suction-velocity.

It has been demonstrated that under the reaction of pulsion with mixed sizes of grains of different specific gravities certain definite positions are established according to diameters. Thus, in the case of quartz and galena, the grain of quartz in equilibrium with a particle of galena was 5.8 times the diameter of the galena grain.

Stated in other words, the results of the pulsion-jig experiments indicate that in order to effect a perfect separation by

pulsion alone, the grains should be sized between the limits of these ratios, which may be distinguished from those of "free-settling ratios" by "interstitial equilibrium factors," or "hindered-settling ratios or factors." It is important to note that they are larger than those obtained by Rittinger's well-known formula. This formula states that in the case of a sphere the uniform velocity under "free-falling" conditions is:

$$v = 5.11 \sqrt{d\,(x - 1.0)}$$

in which,

v = Velocity of fall in meters per second.
d = Diameter of sphere in meters.
x = Specific gravity of sphere.
1 = Specific gravity of liquid (unity in case of water).

Thus, in the case of quartz and galena, if for x the specific gravities of the two minerals are substituted, equating and solving for the respective diameters, a ratio of about 4 to 1 is obtained. It is evident that the reactions occurring during pulsion have resulted in increasing materially the ratios possible under "free-settling" conditions. It seems to me that part of this increase may be accounted for according to Professor Munroe's[10] grain of maximum falling-velocity. He has shown that in a tube the grain of maximum falling-velocity is one having a diameter 0.4 that of the tube. Under the force of pulsion the interstitial channels are constantly undergoing a change in their diameters. A small grain of heavy mineral surrounded by the larger grains of lighter mineral will have frequent opportunities for occupying a channel about 2.5 times its own diameter. No doubt the greater acceleration of the small particle over that of the large one will always aid the separation, as pointed out by Rittinger.

It is evident that the experimental interstitial-factors or ratios obtained in the pulsion-jig are much smaller than called for by Munroe's theory,[11] where, in the case of above minerals, large grains closely surrounded by smaller ones, he obtains a ratio of about 31 to 1 for equal-falling grains.

Whatever may be the theoretical diameter-ratios between two minerals under pulsion, it is an easy matter, as pointed out by Professor Richards,[12] to determine what it is under practical conditions, and the ratios that exist under these conditions on a jig-bed are the ones that most closely concern the mill-man.

[10] *Trans.*, xvii., 645 (1888–9).
[11] *Trans.*, xvii., 650 (1888–9).
[12] *Trans.*, xxiv., 484 (1894).

As a resultant of all the forces acting upon the grains during the pulsion-cycle, a certain definite and distinct separation takes place according to the diameter-ratios of the two minerals. When this point has been reached, further separation, or an increase in the diameter-ratios, is not possible. In order now to remove the small grain of heavy mineral from the large grains of light mineral associated with it, the application of some other reaction is necessary. This force is suction, or, perhaps more properly, the reactions that occur during the suction-cycle.

Under the conditions that exist in a jig-bed, we are dealing with a number of columns of water moving with some velocity relative to the grain. The forces acting upon the grain will be those of the water-currents, of gravity, and of the resistance opposed by the walls of the channel or other grains. The effect of the water-current alone upon the grains may be considered a purely non-selective force. For grains of the same size and shape a given current will exert as much effort upon a particle of galena as upon one of quartz. Any advantage that the small heavy grain has over its larger companion, due to acceleration, will always be a positive force. The resistance offered to the passage of the grain by other surrounding grains will depend upon the relative diameter of the channel and the grain, and the length, shape, and inclination of the channel. If the grains are all the same size and shape, then the mean diameter of the channels will be less than the diameter of any of the grains, and none of them could be carried through the interstitial spaces. Take as an extreme case a column of shot, steel balls, or marbles of the same diameter, they are all absolutely fixed as regards any possible suction-velocity. The same is true, through to a less extent, in rounded particles not all the same size, as well-worn sand, gravel, etc. Again, the possibility of the mass becoming packed is small. Of course, the reason for this is well understood, and is owing to the fact that in these cases the surfaces of the particles are curved, and therefore the points in contact are reduced to a minimum. Under any practical conditions existing in the jig-bed, the particles are not all the same shape or size, and instead of being bounded by curved surfaces they are angular and bounded by planes. This results in neighboring grains having not few but many points in contact, accompanied always by a more or less

wedging action, and therefore jigging under excessive suction-velocity results in a tight bed. The wider the size-ratio the greater the effect, and *vice versâ*. The possibility of applying suction depends upon the ability to maintain within the jig-bed interstitial channels somewhat larger than the maximum grain to be saved. Under the conditions existing on a jig-bed, the effect of increasing the diameter-ratio of bedding and feed, the number of bedding-grains in a vertical column, or thickness of bed, and the character of the bedding-grains themselves, whether they are rough and angular, cubical, or well worn and spherical, is at once evident.

The increased catch secured on a jig-bed over that obtainable by rising current alone, under either free or hindered settling conditions, is due to the reaction occurring during suction. In order that suction may become effective, it is necessary that the reaction of pulsion precede. During pulsion a selection and arrangement takes place; and during suction a destruction of the conditions of equilibrium set up under pulsion, by the removal of the small heavy grain through the interstitial channels into the hutch, results. Suction always supplements gravity, but in a way in which gravity cannot act efficiently—that is, in the movement of grains in channels more or less inclined or crooked, where a particle could easily lodge, although large enough for the grain to move in if vertical. The current moving with high velocity in these spaces serves to move the particle. Pulsion may be said to be the master reaction, while suction is its necessary complement, completing what has been initiated by pulsion. Suction is therefore necessary in jigging all unsized material. Excessive suction with sized material, under practical conditions, would be disadvantageous. With very close sizing on coarse jigs it would not be particularly harmful, but it would be useless. In jigging under any conditions, more or less suction will be of advantage, as helping to save the smaller particles of heavy mineral that otherwise might be carried off with the tailings.

VI. Discussion of Acceleration.

It has been pointed out (Tests 16, 17, 18, 37) that with a very short and quick stroke, but relatively low piston-speed, the ratio of concentration obtained was low. Moreover, the

jig-bed pulsated under the influence of the short, rapid stroke, and did not with the longer one of less frequency, but having the same mean piston-speed in inches or millimeters per second. This was a movement of the grains *en masse*, the bottom pulsating quite as much as the top, and was altogether different from that gentle, selective action observed with proper speeds and frequencies. The jig-bed moved as it would if acted on by a solid piston from below. Thus, by giving many quick sharp blows to the jig-bed, the water-columns have not time to adjust themselves to the increased pressures, except by raising the grain which happens to be in the direction of impulse. In addition to the mean piston-speeds, as derived from Professor Munroe's formula,[13] the element of time during which the impulse lasts should be included. This solid or piston-effect of a water-column can, perhaps, never be entirely eliminated, nor does it seem desirable that it should be. The results show that increased quantities of hutch-work are produced, supplementing suction by keeping the interstitial channels cleared. Since the grains on the bottom are the first to feel the impulse and be raised, it has been shown that true pulsion is diminished, and the important reactions dependent on it diminished. Sharp, rapid strokes, by increasing the piston-effect, promote sifting, and therefore aid suction, but decrease the reaction of pulsion.

VII. Résumé and Conclusions.

Referring to the 13 conclusions of Professor Munroe, quoted in the early part of this paper, it may be said that no experiments have been carried out with the idea of duplicating the work covered by the first six of his conclusions. In the absence of positive experimental data, it may be considered quite out of place to enter into a discussion of them. However, in the light of results of the present investigation, a few observations concerning these first six conclusions may be given. The careful record of so many tests, under the conditions observed by Professor Munroe, seems to cover the field thoroughly.

Conclusions 1, 2, and 3 are undoubtedly fundamental propositions in any system of jigging. To Professor Munroe is due the credit of having first clearly pointed these out and applying

[13] *Trans.*, xvii., 647, 648 (1888–9).

them to jigging. Following, as corollaries, are the formulas given for the velocity of fall of grains *en masse*. The formulas for the falling-velocities of grains *en masse* under the assumed conditions, when applied to piston-speed, have been demonstrated by experiments with the pulsion-jig, the Vezin jig, and the Harz jig to yield satisfactory results.

Conclusion 4 has been noted elsewhere. A grain 0.4 the diameter of the channel will have a maximum falling-velocity, which therefore increases its chance of being saved, and of increasing the interstitial settling-ratio.

Conclusion 5, in the first part, follows, also, from the first three conclusions, and its application is fully demonstrated by Professor Munroe. It seems to me that there is a reasonable doubt about accepting the second part of this conclusion. There is no doubt about this part of it: "The falling-velocity . . . [of a mass of grains] increases or diminishes with the distance apart of the grains," since this is merely a re-statement of Conclusions 1, 2, 3, and the first part of 5. When, however, the balance of this statement is examined—that is, ". . . the velocity of the current necessary to support or raise the mass of grains increases or diminishes with the distance apart of the grains," I believe we are entitled to withhold judgment until it has been shown what these velocities, under the conditions of jigging, actually are. This statement is true if we assume that the velocities supporting or raising the grain are equal to the observed velocities in the free or unobstructed part of the tube; or in practice the piston-speed. But are these the velocities acting upon the grains? Under the conditions obtaining on a jig-bed, the grains occupy a considerable area, and therefore constrict the passage. It is a matter of actual observation that the velocity in the interstitial spaces is much higher than that of the jig-piston. It is the same principle of conducting a given volume of water through a pipe-line made up of, say, a 12-in. and a 6-in. pipe. In the 12-in. pipe the column of water will have a mean velocity of x feet per sec., and in the 6-in. section the velocity has been increased to $4\ x$. Thus, we must be careful not to confuse the falling-velocity of grains *en masse* with the velocity of the water-column actually supporting them during pulsion.

It has been noted by Professor Munroe that spheres falling in tubes have a maximum falling-velocity when the diameter

of the sphere is 0.4 that of the tube; and spheres either smaller or larger than this size fall with less velocity. If the column of water in the tube has a velocity of 0, or is at rest, a solid falling through this water-column will displace a volume of water equal to its own volume as often as it traverses a distance equal to one of its three dimensions. This displaced volume must escape within the interstitial space of tube and body with some velocity, depending on the velocity of the falling body and the ratio of the diameters of the falling body and the tube. If the falling body has a diameter nearly equal to that of the tube, the area of the interstitial space is small, and a low falling-velocity of the body may correspond to a high interstitial velocity of the water-current. Thus, while the velocity of fall decreases as the diameter of the solid approaches that of the tube, at the same time the velocity of the current tending to support it increases. If the body has a diameter equal to that of the tube, any motion of the solid would mean an infinite velocity to the interstitial current, and the body stops. On the other hand, as the diameter-ratio between the solid and the tube increases, the area of the interstitial space increases, and the volume of displaced water decreases, and with it the interstitial velocity, and the body would tend to fall with a high velocity; but the force causing it to fall, its weight, is also smaller, and therefore its ability to overcome the inertia of the liquid, and other resistances, is less, so that its falling-velocity is less. The possibility of interstitial currents depends upon a solid of any diameter less than the tube, and having a specific gravity greater than that of the liquid, and which is free to fall, or resists the motion of a column of the liquid in which it is immersed.

It is evident that if a velocity be given to the water-column in the free part of the tube equal to the observed velocity of fall of the body in the stationary column, then the body will be supported or remain at rest. This velocity of the water in the column is the apparent velocity necessary to support the grain, and some function of the actual velocities supporting it. In jigging, it is not so much the velocity of fall of a mass of grains that concerns us, as the velocity of the current necessary to raise or support them. In jigging, the grains are not free to fall, since they are firmly supported on a sieve, but they are

quite free to move when the interstitial currents are acting in pulsion. When the force due to the velocity of the rising currents is greater than all other forces holding the body at rest, then the body moves in the direction of the greatest forces, and continues its motion so long as the forces are unbalanced. Thus, it has been observed that the particles will be raised to positions higher than at rest during the action of the pulsion-current. The grains in the bed are being raised because each one in motion is seeking a position higher up in the column where the distance between grains is greater, or, in other words, where the interstitial velocity is lower.

Conclusion 6 admits of no doubt.

Conclusion 7 is an axiom as regards the first part. The second part concerns the ratio of equal-falling particles of the pair chosen—namely, quartz and galena. No ratios were obtained in any of the investigations approaching those called for by this theory. Evidently the conditions required by the theory were not present. The conditions assumed were that the fine grains should closely surround the large grain of quartz. It has been observed in all experiments that the large grains quickly settled on the bottom—the smaller and lighter above, whether of bedding or ore. This fact was also pointed out by Professor Munroe in his paper on his experiments with mixed shot.[14] There is but one force that can carry the small, light grain to the top. That force resides in the velocity of the interstitial currents acting during pulsion. Certainly, in the pulsion-jig experiments, where the unsized material was thoroughly mixed, and added practically dry in order to avoid any classification in falling through a water-column, and where very large percentages of the heavy mineral (exceeding 70 per cent.) were used, the above ratios should have been secured. In the case of the above pair it was found that a particle of galena and one of quartz 5.8 times its diameter were in equilibrium. If the conditions called for by the theory were present, and the results not fulfilled, then an examination of the theory is in order. But we have observed above that while the material was thoroughly mixed when added to the tube, the fine, light grains immediately separated from the large, heavy

[14] *Trans.*, xvii., 649 (1888–9).

ones during the first few strokes of pulsion. From this we must conclude that the fine, light grains were not in equilibrium with the large neighbors, and sought positions higher up in the column where they were. When this was found they remained fixed, or were in equilibrium. These ratios have been given elsewhere. For quartz and galena the ratio was 5.8 to 1.

The conditions assumed cannot, under any possible conditions, exist on the jig-bed, and therefore the results that would follow cannot possibly be attained in practice. The conditions would be fulfilled if we caged all the light and heavy grains, and prevented any movement among them; but this is the very condition that we do not want on a jig-bed. It is hardly fair to assume that if in one way or other we are able to keep a mass of mixed grains together, under conditions where the smaller ones cannot escape, therefore the small grain is falling with the same velocity as the large grain. It is in equilibrium by force, not choice; and on the jig-bed we try, as far as possible, to encourage the grains to exercise the latter and not the former.

Possibly if formulas had been derived showing the velocity of the interstitial currents (the currents supporting or raising the grain), and from these, equal-settling ratios were derived, the values would be much less than 31 to 1, and probably close to those obtained in the pulsion-jig.

Conclusion 8 follows from the conclusions 1, 2, 3 and the first part of 5. I can bear testimony as to the practical accuracy of this, since I have calculated many a piston-speed and velocity in the free tube in the pulsion-jig. With the formulas given, it has been shown that with pulsion-jigs, Vezin jigs, Harz jigs, etc., the piston-velocity so calculated suffices to move the grains. Given the size or diameter, and the specific gravity of the minerals to be separated, the jig-piston velocity may be calculated with almost a nicety. It has been shown in the experiments in piston-velocity that a considerable variation is permissible in jigging.

Conclusion 9, the first part of Conclusion 11 and all of 13 are corollaries of the last part of Conclusion 7. Since the conditions assumed for Conclusion 7 cannot exist on a jig-bed, therefore no support is left for 11 and 13, and some other ex-

planation must be given to account for the applicability of the English system. This action has been discussed under pulsion and suction.

Conclusion 10 has been abundantly demonstrated. It might be added that if the theory were applicable little or no suction would be necessary.

Conclusion 11. The last part of this conclusion, concerning the presence of more or less coarse material in jigging very fine material, agrees with practice, since, if not in the feed, a bed is used, which fulfills the conditions. The tests do not cover cases in which any large percentage of feed was less than 0.10 millimeter.

Conclusion 12 accords with all results of practice and experiment, and is therefore another fundamental proposition in jigging.

Finally, to Professor Munroe must be given the credit for having pointed out the fact that bodies fall with less velocity in tubes than in large bodies of water, and for having demonstrated the applicability of formulas based on this fact to obtain correct jig-piston velocities under the assumed conditions. It is to be always understood throughout this paper, that under records of the experiments, where pulsion-velocity and suction-velocity are given, the value expressed in inches or millimeters per second is that of the piston or the water-column in the free or unobstructed part of the jig only, and clearly not the actual pulsion-velocity acting upon the grains during these reactions. One is a function of the other, but under the very conditions obtaining on a jig they cannot be equal.

Comparing the results given by Professor Richards, a part of whose conclusions are quoted earlier in this paper, it will be found that, so far as the experiments may be compared, there is a very close agreement between us. Since we both started from the same experimental basis, on which we were agreed, it is but natural that our conclusions should be in close harmony. This theorem, which forms the basis in every practice of jigging, is all important, and of course is the establishment of the value of the resultant measured by the diameter of grains differing in specific gravity obtained during pulsion alone. This factor represents all that can possibly be expected from every force acting upon the grains of a jig-bed during the time the pulsion-

currents are acting, or while the grains are free to fall. To Professor Richards is due the credit of having demonstrated the value of this resultant as measured by the ratios of diameters. My own researches, carried out differently (see pulsion-experiments), have abundantly confirmed the substantial accuracy of these ratios. When, therefore, Professor Richards says: "The two chief reactions of jigging are pulsion and suction," I see no escape from his conclusion. If we go a little further and say: "The reactions occurring during pulsion and suction are the only reactions of jigging," we have included every force imaginable that can act upon the grains. As pointed out above, the resultant of all the forces acting upon the grains during pulsion is given by the interstitial or hindered-settling ratios as determined by Richards and myself. The resultant of suction cannot be separately determined, apart from that of pulsion. There is no determinable resultant of suction as measured by a ratio or factor.

Summarizing some of the principal points brought out in this investigation, I believe the following may safely be accepted:

(1) The pulsion-reaction is by far the most important one in the process of jigging. During this period, with sized grains of different specific gravities, with proper pulsion-velocity, the separation between them will be complete. The size-limit is indicated by the hindered-settling ratio. If the minerals are not sized, or above these ratios, the separation cannot be complete, but a definite arrangement will result. The positions of equilibrium will be attained when the above ratios of diameters are attained, after which further separation by pulsion is impossible.

(2) Suction due to the movement of water-columns supplements gravity. Resisting the sum of these two forces is the resistance of the walls of the tube through which the grain must pass. The reaction, as a whole, must therefore be a resultant. The chief component is the force of the water-columns, which are purely non-selective, but act with equal intensity upon all particles of the same shape and size, regardless of their specific gravity or weight. Any advantage that the small heavy grain would have over a large light one would, of course, appear in the resultant tending to carry it to the hutch. The effect of the forces opposing the movement of the grain depends upon

the character of the grain, and the conduit through which it is supposed to pass. Under any condition, the diameter of the grain cannot be greater than that of the conduit. If the channels are inclined, or crooked and zigzag (the condition obtaining on a jig-bed), the particles will more easily lodge against the sides of a tube large enough to pass through if the tube were vertical, but under the force of gravity they remain at rest. The rapidly descending water-currents passing through these channels easily carry the grains along. Thus suction, due only to the moving columns of water, constitutes a powerful impelling force to carry through the interstitial spaces those particles which under the force of gravity alone cannot move. Suction is, therefore, a necessary complement to pulsion in the jigging of all unsized material, and generally valuable in jigging under all conditions.

(3) From the observations under (2) it is clear what effect the bedding will have upon the result. Any part of the bedding or ore-column remaining fixed during the pulsion-cycle must be looked upon merely as a mass of very irregular tubes, of length somewhat greater than the thickness of such part, owing to their inclination, since they are mostly inclined. To that extent they are only an extension of the jig-sieve. The result of thickening or thinning the bed, or of increasing or decreasing the size-ratio between bedding and feed, is evident. This assumes, of course, that the largest particle of feed is smaller than the sieve-aperture, and always the bedding-grain must be larger than the sieve-aperture. It is evident, too, that the shape of the bedding-grain will have a marked effect. Grains that are more or less equi-dimensional, as galena, etc., will form a more open bed than one of antimony, which breaks into long pencil-shaped grains. Finally, of course, if the bedding is in use long enough all grains become worn and spheroidal. Any part of the bedding free to pulsate is to be considered as part of the ore-column, and is amenable to all the conditions applying to this reaction.

(4) The effect of very rapid acceleration, amounting to a shock or blow to the bottom of the jig-bed, is an important factor. Its effect is to accelerate the work done by suction, and render a larger catch possible with a low mean piston-velocity. The pulsation of the jig-bed due to this force and

that taking place under the regular interstitial velocity should be distinguished. One sifts, the other separates.

(5) The results of the many experiments, in which the piston-speeds during the pulsion and suction were not the same, seemed to show that only by properly balancing the two are the best results attained. It has been generally noted that the eccentric, giving equal mean velocities, yields about as good results as any of the accelerated strokes. This observation applies only for the size-ratio used in the tests, and it is not safe to speculate what the results would be for other sizes.

(6) While the use of the jig for the treatment of material sized between wide limits is possible and practicable, still the advantages that are bound to follow where a more or less perfect sizing has preceded cannot be denied. It must be observed, that in the English system itself, when the hutch-products of one jig are treated on another we are using sizing.

(7) The more general application of the English system, or the use of the jig in the treatment of unsized material instead of the hydraulic classifier, seems to be clearly indicated. This has been recognized in some quarters, but a wider use than has hitherto been accorded it appears to hold out favorable inducements. This seems to be a field eminently suited for the English methods of jigging—one that is not and cannot be filled by the Continental system.

(8) The arguments that have been advanced for the adoption of the English system on the ground that equal-settling ratios, many times larger than those obtainable under free-settling conditions, exist on the jig-bed, are not tenable. These hypothetical ratios cannot possibly exist on a jig-bed.

In conclusion, I must acknowledge the great help and many suggestions derived from the works of Professor Richards and Professor Munroe; to the latter personally, I owe cordial thanks for numerous timely suggestions. I have also had the benefit of his valuable criticism in the construction of the laboratory-jig, with which many of the experiments were made.

A New Theory of the Genesis of Brown Hematite-Ores; and a New Source of Sulphur Supply.

BY H. M. CHANCE, PHILADELPHIA, PA.

(Chattanooga Meeting, October, 1908.)

STRETCHING from New York southwestwardly to Georgia is a great range of hills and mountains consisting of pre-Palæozoic schists, slates, and gneissic and granitoid rocks, known locally by many different names; in Pennsylvania as the South mountains, the Berkshire or Durham and Reading hills, and in Virginia and farther south as the Blue ridge. To the northwest of this range lies the "Great Valley," underlain by Lower Silurian limestones, and known as the Lehigh valley, Cumberland valley, Shenandoah valley, valley of Eastern Tennessee and Georgia and by other local names, but all these are parts of one valley, geologically and geographically continuous from New Jersey to Georgia.

Lying upon and along the northern flank of this range of mountains, and dipping northwestwardly under the limestone of the valley, is the Potsdam formation (Primal or No. 1 of the Pennsylvania Geological Survey), which consists of sandstone or conglomerate overlain by slates. The latter are sometimes arenaceous and occasionally are replaced in part by sandstone. This formation varies from a few feet to several hundred feet in thickness.

At the outcrop the slates are completely decomposed, oxidized or disintegrated, forming a belt of clay several hundred feet wide occupying the interval between the outcroppings of the sandstone and limestone. This clay-belt includes a more or less continuous series of iron-ore (brown hematite) deposits, which have been, and still are, extensively worked.

The belt is more or less continuously mineralized with iron throughout its length, and hundreds of large deposits of ore have been developed and worked, producing many millions of tons of ore.

The extent of this ore-belt has been dwelt upon by many observers, including miners, engineers, and geologists, all of whom agree as to the enormous quantity of ore available. The following quotations are of interest in this connection.

Prof. W. B. Rogers, in *Geology of the Virginias*, speaks of this range as "continuous and inexhaustible." In the *Tenth Census of the United States*, 1880, the deposits are described as "continuous beds." Maj. Jed. Hotchkiss says their "variations from point to point are no greater than those found in coal." Prof. W. M. Fontaine speaks of them as "overlapping lenticular deposits—for long distance—extending along a definite plane." Prof. J. P. Lesley, *Proceedings of the American Philosophical Society* (1873), describes these ores as "geologically continuous," and in subsequent writings as "perfectly well-defined and continuous strata"—"in parts of the belt ore exists in millions of tons"—"the quantity attainable has no easily assignable limits and may furnish supplies for centuries."

It therefore can be stated with confidence that this range is one of the most extensive and continuous mineral-deposits or series of mineral-deposits known in this or any other country. The ore, however, is not of the best quality; it carries a large percentage of silica, contains from 10 to 14 per cent. of water, has an objectionably high phosphorous-content (always above the Bessemer limit), and consequently does not command a high price at the furnaces. As the clay-stripping is usually heavy and increases rapidly with depth, the mines are shallow open pits, seldom more than 40 ft. deep, and when, from increased depth and cost of stripping, the ore can no longer profitably be mined, the workings are abandoned and new pits are opened at some other place. Underground mining has been prosecuted to some extent at a few localities, but rarely with profit, such operations usually being abandoned before reaching permanent water-level. Hence, little is known concerning the extension of these ores in depth.

At an early date it was known that the slates (clay) in which these ores occur contain more or less disseminated pyrite. Upon this fact the generally accepted theory of the origin of these ore-bodies has been predicated.

With few exceptions geologists and miners have adopted the theory that these "mountain" ores are precipitations from downwardly percolating waters which previously had leached the iron from the disseminated sulphides of the Potsdam slates or of the older underlying (pre-Palæozoic) formations. The

publications of the various State geological surveys and the writings of geologists who have studied these deposits all tell practically the same story. A few, however, have held that the source of the iron was in sulphides or carbonates existing in the overlying but now eroded Silurian and Devonian formations, but this hypothesis has had few adherents.

The possibility that extensive and more or less persistent beds of iron sulphides were the source of these ores does not seem to have been seriously considered. That this theory has not been accepted is proved by the fact that shafts or bore-holes have not been sunk to develop the deposits, for active search has been made for pyrite in almost every locality in the eastern States where its presence is suspected.

In an article by Prof. T. C. Hopkins,[1] entitled the Cambro-Silurian Limonite Ores of Pennsylvania, a bibliography especially referring to the valley limestone-ores is given, including a summary of the views of each writer upon the genesis of these ores. Without reproducing this matter in detail, it is sufficient to say that the following list of authors describe these ores as secondary precipitations from waters carrying the iron in solution:

E. R. Benton; Gustav Bishop; E. V. d'Invilliers; A. L. Ewing; W. M. Fontaine; Persifor Frazer, Jr.; A. A. Julien; J. D. Kendall; James P. Kimball; J. P. Lesley (?); J. S. Newberry; R. A. F. Penrose, Jr.; J. B. Porter; F. Prime, Jr.; H. A. Rogers; T. C. Hopkins. To this list might be added the names of many others who have written upon this subject, including F. L. Garrison and C. W. Hayes.

A few hold that the deposits are concentrations, or the oxidized remains of carbonates, or the remnants of original limonite deposits, *in situ*.

Among these are J. D. Dana; J. A. Harden; R. M. S. Jackson; J. P. Lesley; B. S. Lyman; and F. J. H. Merrill.

A still smaller number have suggested that the ores result from the oxidation *in situ* of pyrites, iron carbonates, or other ferriferous minerals.

Among these writers are some of the earlier authorities: C. U. Shepard; T. Sterry Hunt, and others; but they were seemingly uncertain of their ground,

[1] *Bulletin of the Geological Society of America*, vol. ii., pp. 475–502 (July, 1900).

and as they failed to elaborate their views by the presentation of convincing arguments or conclusive evidence, the theories which they advanced were not accepted by others, and have been relegated to the background by more recent writers.

More than 20 years ago I became convinced that these mountain-ores were "gossans;" that is, the oxidized remains of pyrite-deposits, practically in place, and therefore underlain by undecomposed pyrite in place. Being at that time engaged in mining iron-ore from these deposits in Pennsylvania, I became satisfied that the generally accepted theories were not supported by the observed facts. That other geologists have not reached the same conclusion may be due to lack of opportunity for close observation, such as is open to those actually engaged in mining. The miner, who day by day observes the varying peculiarities of the ore he is working, sees many things hidden from the casual observer, and is thus better equipped to solve problems of structure and origin than he whose opportunities for observation are restricted to occasional and hasty visits. The conclusion that these ores are gossans was not based solely upon the occurrence of pyrite in isolated masses in clay (partly decomposed or oxidized) or in the interior of lumps or masses of limonite-ore, but was based largely upon the character of the ore-bodies and of the materials in which they are incased.

Most of these ore-deposits consist of a main ore-body of massive or solid "lump" ore, having many of the characteristics of a well-defined shoot, vein, or bed; that is, reasonable persistence and continuity lengthwise, a well-defined dip, and perhaps the lenticular form of a shoot with a definite pitch. Accompanying this bed or vein of solid (lump) ore, which contains comparatively little clay, there may be considerable deposits consisting of ore-particles disseminated through a relatively large quantity of clay. These disseminated deposits are of irregular size and location, may be found on either side of the vein, and are of extremely irregular conformation. Ore of this character is known as "wash-ore."

As the decomposed slates (clay) are nearly impervious to water, it seems quite unlikely that the main bed or vein was precipitated from percolating waters, for to permit such deposition it would be necessary to provide a porous or soluble stratum, such as limestone, which by dissolving might provide

space for the deposition of ore and open channels for the circulating waters. But there is no evidence of the presence in these slates (clays) of any such stratum. How then could such waters circulate or flow through these clays or slates, and why should they preferentially select this formation in which to circulate and to precipitate the dissolved iron, when nearby was the great body of permeable valley- (Silurian) limestones providing innumerable channels for their escape? Further, it seems impossible by this theory to account for the masses of disseminated ore, the particles of which are sealed and isolated in clay impermeable to water. This clay, moreover, is frequently white and quite free from iron, which could not well be the case if for long periods it had been saturated with water heavily charged with unstable salts of iron.

These physical contradictions alone are sufficient to disturb faith in the generally accepted theory of chemical precipitation from ferriferous waters. Furthermore, if this theory be true the ore should frequently be found to terminate abruptly in depth, but this it rarely if ever does. I know of no instance in which the ore so terminates.

Having dismissed this theory, and having observed the occasional presence of pyrite in the lower workings, I became convinced that these ores are "gossans," and that the sulphide-deposits, of which they are the oxidized remains, exist beneath them and probably extend to great depth. As in many cases the surface ore-bodies are very large, often being almost continuously exposed for thousands of feet, and being from 10 to 50 ft. thick, the sulphide beds, veins, or shoots from which they originate should be of commensurate size, and capable of yielding enormous quantities of pyrite.

Deductions based upon the structure of the inclosing rocks and upon the physical characteristics; that is, the dip, strike, continuity, etc., of the veins or deposits as already outlined, therefore tend to show that the hypothesis of deposition from percolating water is untenable.

Before proceeding farther, a description of one shoot typical of this class of ore-bodies may be useful. Such a deposit is located at Easton, Pa., along the foot of the north flank of the South mountains, and has been worked for many years.

The vein is opened by a number of shafts through which

mining has been carried on for about 4,000 ft. in length and to a depth of 270 ft. The shoot is from 5 to 10 ft. thick. There is, of course, the usual disseminated limonite in some portions of the walls; however, in places this is missing, and the line of demarcation between the lump-ore and the walls is well defined.

From the surface to about 150 ft. in depth the ore dips into the hill; that is, to the south, but it then straightens up and dips to the north, very prettily illustrating the effect on the dip of the "creep down hill."

In addition to the main vein, there are several smaller parallel veins which, as depth is attained, seem to approach the main vein. This convergence probably results from an increase in the thickness of the parting clay near the outcrop, caused by the "creep down hill" and settlement.

The ore at this mine is a hard compact limonite. It contains the usual high percentage of silica. The presence of pyrite has not been noted either in the form of isolated masses or lumps (sulphur balls) or incased in ore. The lower limit of the zone of oxidation has been deeper in this locality than in many parts of the ore-range, doubtless owing to the proximity of the Delaware river, and the height of the outcrop above water-level.

In the lower levels of these mines the ore is fissured and the fissures are usually water-courses, through which waters may readily be carried down to the zone below, in which oxidation is probably now progressing, at or below permanent water-level.

Intermixed with the limonite in the deeper workings some very pure carbonate ore is found, which will be referred to in discussing the chemical reactions by which the peculiarities of these ores may be explained.

While I have attempted to show that upon purely theoretical grounds the hypotheses popularly accepted as explaining the genesis of these ores are untenable, some positive evidence as well as inductive reasoning will now be presented to show that the ores are "gossans;" that is, are derived from the oxidation of pyrite and pyritic material *in situ*.

As yet there are but few places where pyrite has actually been found in quantity, and until quite recently these have escaped attention, but at two localities with which I am familiar

notable quantities of pyrite have been found. The first of these is about 2 miles east of Emaus, Lehigh county, Pa., and the second is about 1.5 miles west of Alburtis, near the Lehigh-Berks county line. At the former locality the ore has the structure and appearance of sandstone, and even when it consists almost wholly of pyrite (the silica or insoluble residue often being no more than from 15 to 18 per cent.) the granular structure of the sandstone and its lamination and bedding are as perfectly preserved as in specimens which are merely pyritic sandstone. All grades of materials are found at this place, ranging from sandstone with little or no pyrite up to pyrite ore as above described, containing about 84 per cent. of pyrite. (The term pyrite has been and will be used throughout this paper to indicate either pyrite or marcasite or a mixture of both. No determination has yet been made which justifies me in specifically indicating the species, and the term pyrite is therefore used conveniently to mean iron bi-sulphide, FeS_2.)

The pyrite consists almost entirely of fine grains which rarely show crystalline faces, and generally are small enough to pass through a 20- or 30-mesh screen; a large portion passes through an 80-mesh screen, and a considerable percentage is of still finer texture.

The extraordinary preservation of the lines of sedimentation and the integrity of the individual sand-grains seem to refute the hypothesis of mineralization subsequent to the deposition of the sandstone, for this would require in some cases the removal of 84 per cent. of the silica, its place being taken by pyrite. The residual sand-grains, however, do not appear to have been subjected to the action of solvents, and, further, a replacement of this nature would in all likelihood be accompanied by an obliteration of the granulation, bedding, and lamination.

Hence, it appears that the pyrite must have been deposited with the sand at the time of deposition, either by mechanical transportation and concentration (as magnetic sand is transported and concentrated in littoral and other deposits), or by precipitation in lagoons, bays, or channels. The physical appearance of the ore, and the size and conformation of the individual pyrite-grains, plainly indicate that the pyrite is an original constituent part of the sandstone.

That some of the iron-ore is directly derived from pyrite of this character is proved by numerous specimens, parts of which are completely changed into iron-ore of typical character, preserving the lamination of the original sandy pyrite, the remainder being entirely unchanged pyrite.

At the second locality pyrite has been found in three shafts, and some drifts and cross-cuts have been driven which are partly in pyrite and partly in iron-ore. At this mine the ore differs from that at the first-described locality in that the gangue is clay instead of sand; the pyrite having doubtless been deposited in beds of mud or clay which subsequently became slates and by recent decomposition have reverted to clay.

The pyrite is largely disintegrated, and much of it is an impalpably fine but pure pyrite slime, frequently showing not more than from 3 to 5 per cent. of foreign matter. Some hard (solid or massive) pyrite is associated with the soft material, and some of this also is very pure, showing by analysis more than 52 per cent. of sulphur; that is, it contains not more than 2 per cent. of foreign matter. In addition to ore of this character, some layers of bluish-black clay were passed through containing much finely divided pyrite—evidently a partly decomposed and disintegrated pyritic slate.

With deeper development this pyrite will doubtless be found massive and in bodies of considerable size inclosed in pyritic slate.

It is interesting to note that where the sediments are fine-grained (mud) the pyrite is also fine-grained, and where the sediments are coarse (sand) the pyrite is correspondingly coarser. This peculiarity furnishes additional evidence that the pyrite is a mechanically transported sediment, possibly derived from the erosion of older eruptives. It is, however, possible that this similarity between the relative coarseness or fineness of the pyrite and the gangue is merely an accidental coincidence, or is due to hydraulic classification by tidal currents.

While no specimens were obtained from this mine showing pyritic slate partly changed to iron-ore (similar to the sandy ore already described), it is practically certain that such slaty pyrite exists, changing by oxidation to iron-ore which preserves the structure of the slaty pyrite from which it was derived, for mines lying in the slates some distance from the Potsdam sand-

stone frequently produce iron-ore which is of this slaty character, and such ore is said to have been found at this mine when

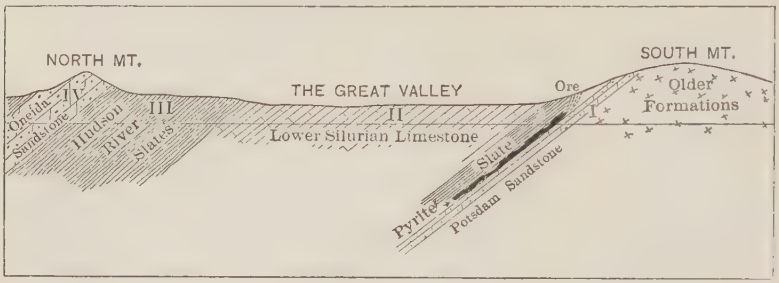


FIG. 1.—TYPICAL SECTION ACROSS THE GREAT VALLEY.

it was worked by open cut as an iron-ore mine. Ores of this character attracted the attention of Lesley 40 years ago, who

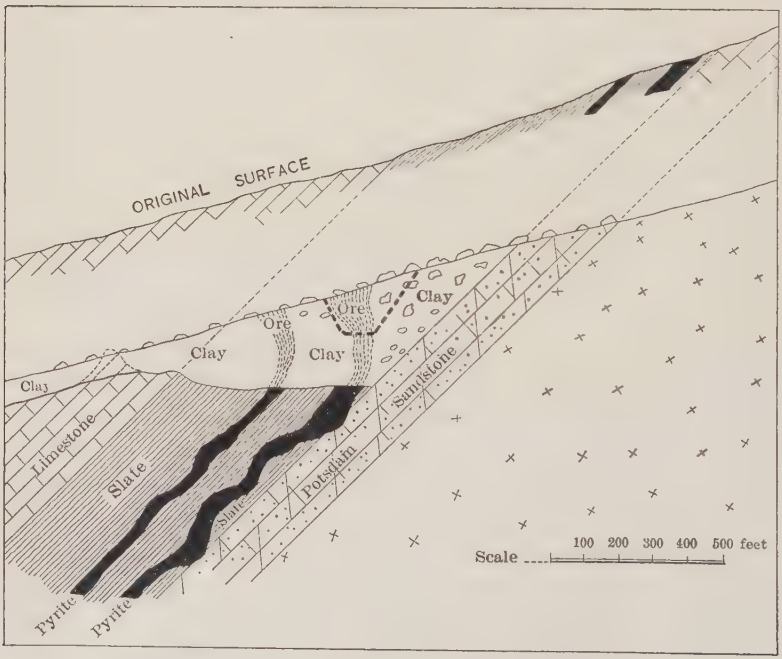


FIG. 2.—SKETCH SHOWING RESULTS OF OXIDATION OF PYRITE TO IRON-ORE, DECOMPOSITION OF SLATES, AND "CREEP DOWN HILL."

described them as rock changed to ore, but he assumed that the percentage of iron was increased by the removal from the slate of its soluble portions and not by the oxidation of pyrite.

He emphasizes the fact that these ores preserve all the characteristics of the rocks from which they were derived, but evidently had no thought that they were alterations of pyrite in place.

The hypothesis of mineralization of the sandstone or slate subsequent to deposition, resulting in pyritic pseudomorphs after sandstone or slate, requires either that the pyrite was deposited in cavities originally existing as such or resulting from the removal of soluble parts of the rock, or that a true pseudomorph was formed by a replacement, grain for grain, of a certain portion of the rock-mass. As the sandstones and slates when free from pyrite are homogeneous sand- or mud-sediments, and do not consist of an admixture of two or more materials, the removal of one of which might produce the required cavities or pores, and as sandstone or slate consisting of one-third solids and two-thirds voids is inconceivable, the pyrite could have been introduced only by replacing the rock, grain for grain and particle for particle, thus producing a true rock pseudomorph.

Unfortunately for this contention, microscopic examination fails to show any sand-grains partly replaced by pyrite, nor do any parts of the specimens show a complete removal of the sand, which would be expected if the sand had been removed by an active solvent of silica.

As the presence of the pyrite cannot be explained by either of these hypotheses, it seems quite certain that it was deposited, either mechanically or by precipitation, with the sediments of sand or mud of which these sandstones or slates are composed.

If we inquire how such deposition or precipitation could be effected, it will be easily understood how the pyrite may have been concentrated and distributed by waves or currents, and at the same time hydraulically classified into coarse material depositing with the sands and fine material depositing with the muds.

It is, however, entirely possible that the granular condition of the pyrite is due to crushing; that the pyrite was deposited by precipitation, subsequently became massive and solid, perhaps somewhat crystalline, and was reduced to a granular condition by destructive pressure. This view is supported by microscopic examination of the pyrite, the grains of which are

rather sharp, do not show much evidence of wear, and present few crystal-faces.

The reduction and precipitation of iron bi-sulphide is usually attributed to organic matter, or to the action of growing vegetation. In this way pyrite is supposed to be precipitated in marsh and peat-bogs, and the pyrite contained in coal-seams may reasonably be accounted for in the same way. It does not seem improbable that a profuse growth of marine vegetation (sea-weeds) may have been the agency through which pyrite in quantity was reduced and precipitated while the sands and muds were being laid down, and possibly the granular form of the pyrite may be explained by assuming that granules of pyrite formed on floating sea-weed, and that as this died and decayed the pyrite granules became detached and settled into the sand- or mud-sediments. Possibly other conditions may have caused a reduction and precipitation of iron as bi-sulphide—conditions which our present knowledge does not enable us to imagine or describe.

Having shown that the ores are probably gossans resulting from the oxidation *in situ* of pyrite, pyritic slate, pyritic sandstone, or of pyrite with a clay gangue, or with a sand or sandy gangue, it is now proper to inquire whether the oxidation of such material will produce iron-ores of such quality and kinds as are found in these deposits.

It may be assumed at the outset that oxidation of pyrite below the surface does not proceed in exactly the same way, nor yield the same products, as oxidation in the open air, for while oxidation with full exposure to the air frequently results in freeing sulphur, the oxidation of pyritic deposits underground rarely gives this result.

From the experimental work of Dr. A. P. Brown,[2] upon the oxidation of pyrite and marcasite in an oxidizing environment, and of Dr. H. N. Stokes[3] with solutions of alkaline carbonates from which the free oxygen had been expelled, it seems quite possible that in either acid or alkaline environment, in either the presence or the absence of free oxygen in solution, complete oxidation of pyrite or marcasite may occur at normal or at very

[2] *Proceedings of the American Philosophical Society*, vol. xxxiii., pp. 225 (1894).

[3] *Bulletin* No. 186, *U. S. Geological Survey*. Also quoted in *Trans.*, xxxiii., 753 (1903).

moderately elevated temperatures and pressures such as may exist at or within a few feet of the surface; the products being ferric oxide, ferric hydrate, and ferrous sulphate in varying proportions, and sulphuric acid as a constant and necessary product when all of the sulphur is completely oxidized. Under favorable conditions possibly all of the sulphur may be oxidized directly to sulphuric acid, the iron being oxidized directly to ferric hydrate, but the more certain reaction which produces ferrous sulphate—immediately precipitated *in situ* as carbonate (quickly changing to ferric hydrate) or as ferric hydrate—and sulphuric acid may perhaps more nearly agree with present chemical theories; but either reaction enables us to explain satisfactorily the genesis of the various forms in which the brown hematite-ores occur.

The most interesting and suggestive result of Dr. Brown's researches is found in the experiments with a cupric sulphate solution. At a temperature of about 200° (392° F.), corresponding to a gauge-pressure of about 220 lb. per sq. in., this solution completely decomposed the sulphide. In other words, as the solution of a neutral salt, $CuSO_4$, in the absence of free oxygen, at very moderate temperature and pressure decomposes both pyrite and marcasite, other salts may likewise assist in decomposing the water, for under such circumstances hydrolysis must be necessary to oxidation.

The reactions of the oxidizing process and the chain of succeeding reactions will depend upon or be effected by the character of the gangue. In all cases, that is, whether the gangue consists of sand, sandy clay, or clay, the oxidizing processes in the first stage are identical, producing ferrous sulphate or ferric oxide or hydrate, and sulphuric acid. The ferrous sulphate may immediately be precipitated as carbonate or hydrate, or some of it may be carried away in solution. When the gangue contains bases that may be acted upon by the acid these are decomposed, and the solution then contains sulphates of aluminum and of the alkalies and alkaline earths, and free silicic acid, all of which may be removed and carried off in solution. In some cases the silicic acid set free, uniting with the ferric oxide or hydrate, may produce iron silicate, which often occurs in large quantity as a hard, flinty, brown or "liver-colored" ore.

If the slate or sandstone has been subjected to pressure and movement sufficient to shatter the rock or to produce a large

number of cleavage-joints or planes, the oxidation naturally commences along these joints and planes, which become channels through which the oxidizing waters find access to the body of the pyritic mass. When a fragment of such pyritic material is attacked oxidation will commence from the outside, changing the outer layers of the fragment to limonite, and producing sulphuric acid and perhaps some ferrous sulphate. The latter may be removed by the circulating waters. The sulphuric acid attacks such portion of the gangue (such as clay) as is decomposable by acid, dissolving it as aluminum sulphate and silicic acid, which will also be removed by the circulating water. As the process continues the film of limonite forming the outside of the lump or fragment grows by successive additions of limonite, which adhere to the inner side of what soon becomes a cavernous lump of ore. The resulting limonite may occupy less space than the original pyrite, because of the removal of some of the iron as iron sulphate and of some or all of the clayey gangue. The process having continued until all of the pyrite is oxidized, a hollow mass of limonite—known as a "bomb" or "bomb-shell"—remains. If the pyrite contained a larger percentage of clayey gangue than could be decomposed and removed by the sulphuric acid produced by oxidation of the sulphur, the residual excess of undecomposed clay remains in the bomb, filling or partly filling it.

If the original pyrite had a clayey gangue with some intermixed sand, the clay may have been entirely removed and the sand left behind, and if the gangue consisted wholly of sand all of the sand would remain in the interior. Thus we have conditions satisfactorily explaining the formation of bomb-shell ore, and also accounting for the fact that some of these bombs are empty, some contain loose sand in large or small quantity, and some contain clay or ocher. Further, the quartz-crystals or chalcedony lining the interior of some of these bombs may be deposited from the silicic acid set free by decomposition of the clay; for if the bomb be without cracks or openings, while the water may transfuse through its walls, the colloidal silicic acid may be retained. I have recently presented a more complete discussion of this subject in a paper entitled, The Origin of Bombshell Ore.[4]

[4] *Proceedings of the American Philosophical Society*, vol. xlviii., p. 135 (1908).

Waters containing ferrous sulphate derived from the oxidation of these pyrite-deposits readily account for the origin by precipitation of iron carbonate (siderite), and to such cause we may assign the occasional occurrence of masses of siderite in the deeper workings of brown hematite-ore mines, and also some of the various forms of stalactitic or "pipe" ore.

Flake- or sheet-ore may likewise be formed by deposition as carbonate or by direct precipitation as hydrate, in cracks, joints, and fault-planes, but the flat or flaky fragments so common in these ores are doubtless derived more frequently from laminæ and layers of pyrite or of pyritic slate or sandstone by direct oxidation of these layers *in situ*. The laminated structure of the original rock is often faithfully preserved in such ore, which is accompanied by an abundance of similarly oxidized slate-fragments, too lean to make merchantable ore and which must be removed by jigging or hand-picking.

The presence of manganese was disclosed by an examination of the unaltered pyrite of some specimens, and sufficient quantities were found to account for the presence of manganese in iron-ores of this character. Whether the manganese exists as sulphide or as oxide has not been determined. It is hoped shortly to make some special examinations to determine the condition in which the manganese exists in the unaltered pyrite.

The chemical reactions resulting in the production of aluminum sulphate, iron sulphate, and silicic acid, all of which may be removed in solution, readily account for the deposits of bauxite and siderite which frequently are found in limestone, or associated with limestone, or in a limestone environment, in regions not far removed from important deposits of brown hematite-ore. Thus, waters carrying salts of aluminum, iron, and silicic acid in solution, upon uniting or merging with waters containing alkaline carbonates—that is, carbonates of the alkalies or alkaline earths—or upon contact with such carbonates, would be expected to precipitate aluminum hydrate (bauxite) and ferrous carbonate (siderite); this siderite upon oxidation forms with the silicic acid the impurities common to commercial bauxite-ores.

The occurrence of ocher with these and other brown hematite-ores is readily explained; for microscopically fine pyrite (slime), if deposited with clay, ultimately hardening to pyritic

slate of extremely fine texture, would form a product which upon oxidation and decomposition might readily produce ocher similar to that which everywhere accompanies this and other similar formations, such, for instance, as the Weisner formation of Alabama.

This theory of the genesis of brown hematite-ores of the Primal slates may be extended to include also the origin of what are known as the Oriskany ores, the "block" ore of the Middle Silurian (No. IV.) sandstone-formation, and may likewise be applied to the limestone ores of the Great Valley, and of other Lower Silurian (I prefer the nomenclature of the older geologists, and use the term Lower Silurian in preference to Ordovician, because it will be better understood by a large number of readers) limestone valleys. The theory as applied to limestone ores may be briefly stated.

Overlying the limestone of the Great Valley is the Hudson River and Utica slate-formation (No. III. of the Pennsylvania Survey). At the base of this slate-formation there is frequently found a pyritic band of slate of considerable thickness. All of the slates from the base of the formation up to the Oneida sandstone are more or less pyritic, and many geologists have attributed the limestone ores to decomposition of the pyrite of these slates, the leaching-out of the iron as iron sulphate and its reprecipitation in the underlying Silurian limestone. While it is evident that large deposits of iron-ore could be formed by such oxidation, transportation, and precipitation, the theory entirely fails to account for the great masses of clay in which these ores exist, for the general occurrence of "bomb-shell" ore (a very large portion of all such ore-deposits, especially the "wash-ore," consisting of bombs or of the fragments of broken bombs), or for the presence of inter-laminated beds of unaltered pyrite at the bottom, or in the deeper workings of some of these mines.

A theory which seems more in harmony with existing and observed facts, and which accounts for the irreguar distribution of these ore-bodies throughout the whole limestone region, will now be stated.

This theory presupposes the existence in the slates overlying the limestone of considerable deposits of pyrite, or of slates sufficiently rich in pyrite to form bombs of ore as already

described. At a time when the original surface was many hundred feet above the present surface, the limestones of the present valley were covered by these slates containing irregular deposits and masses of varying richness and composition as regards their pyrite-content. Oxidation of these masses of pyrite or pyritic slate, proceeding from above downwards, ultimately would leave a mantle of clay resulting from the weathering of the slates, and ore-bodies imbedded in such clays, resulting from the decomposition of the pyritic portions of the slate. Meanwhile, the underlying limestone had been furnishing drainage, and the percolating waters had been dissolving and carrying away large portions of the limestone, especially from beneath those portions of the slates in which active oxidation of the larger pyrite-deposits was progressing, and great caverns and underground water-courses had formed, which were largest and most numerous beneath the larger iron-ore bodies which were being formed by oxidation of the overlying pyritic slates. The general lowering of the level of the whole limestone surface upon which these deposits rested, and the occurrence of great sink-holes in which the ore-bodies gradually sank, prevented the removal by erosion of many of these ore-bodies. If sufficient time be given for erosion to lower the whole valley to its present level, removing in course of such erosion most of the clay mantle, and permitting the further sinking of the residual ores into caverns or sink-holes, the resulting conditions would closely resemble those which now characterize these limestone-ore deposits.

In many cases the ore-bodies have been carried down into the limestone so rapidly that the lower layers of pyrite and slate have not been completely oxidized or decomposed, having probably been continuously at or below water-level, and some unaltered pyrite is found interstratified with limonite bands and slate in the deeper portions of such deposits. Illustrations of these conditions may be seen at mines near Breinigsville and Trexlertown, where merchantable pyrite has been mined and shipped, and also near Fogelsville and in the Saucon valley, a few miles west of Friedensville. All of these localities are in Lehigh county, Pa. It is, however, quite improbable that these limestone iron-ore mines will ever become an important source of pyrite, for they contain only the fragmental residues of the

original deposits, existing under conditions unfavorable to low mining cost.

It is of course apparent that this theory may explain the genesis of brown hematite-ore existing in other limestone formations than those of Lower Silurian age.

If it be asked what quality of pyrite the Primal deposits may be expected to furnish, the question may in part be answered by examining the iron-ores which have been formed by the oxidation of these deposits. These ores rarely contain less than 10 or 12 per cent. of silica, and this quantity of silica may be anticipated in the pyrite, which latter would then contain about 88 or 90 per cent. of pyrite, equivalent to 47 or 48 per cent. of sulphur. By selecting localities where the iron-ore has been of rather better grades, it may fairly be inferred that pyrite carrying from 45 to 50 per cent. of sulphur can be mined.

That the quality of the iron-ore furnishes a reliable index to the character of pyrite from which it has been formed, is, however, true only of those ores having a sandy or siliceous gangue, and does not apply to localities in which the gangue is argillaceous, for much of the clay of such ores may have been decomposed by the sulphuric acid resulting from oxidation of the pyrite and removed by leaching. It may therefore be assumed that, as the argillaceous ores have been somewhat purified by this process, the quality of the iron-ore does not indicate the percentage of gangue in the pyrite from which it was derived.

Another question of prime importance in determining the value of these deposits as a source of pyrite-supply, is the depth to which oxidation extends. Probably the limit of oxidation will be at or near the water-level, or drainage-level, in any particular region, but it is evident that at many points oxidation has ceased at less depth, and owing to comparatively recent changes, by faulting, in the level of the surface in certain localities, undecomposed pyrite exists far above the permanent water- or drainage-level of the surrounding country. This question is one that can be determined only by a study of the local conditions, and perhaps in many cases actual development will be necessary before the depth to workable pyrite can be determined. The method of development obviously may be either by shaft-sinking or by bore-holes. Owing to the frequent overturning of the dip of the oxidized portion of these ores, the

proper location of such bore-holes may be a matter of nice judgment and of some difficulty.

The quantity of pyrite available in these deposits is probably so large that it would not be extravagant to believe it sufficient to supply the world for centuries. This opinion is based upon the enormous quantity of iron-ore which has been mined from the oxidized outcroppings of these deposits, amounting to many million tons.

While the Primal-slate (Potsdam) ores have been described only as they exist on the flank of the South Mountain-Blue Ridge range, extending from the Delaware river southwestwardly to Georgia, the same or precisely similar ores occur in New Jersey, New York, and in some of the New England States, as well as in Alabama and Tennessee. In any or all of these States the search for pyrite may therefore be undertaken with a reasonable expectation of developing workable deposits. In prosecuting such search, it is of course desirable to select those localities in which oxidation has penetrated to least depth beneath the surface, and where the character of the iron-ore indicates pyrite of fair quality.

Should these anticipations be fulfilled, the development of these ranges of pyrite-bearing slates, both those underlying and those overlying the Lower Silurian limestone, will add a great source of wealth to the United States, bringing about the development of chemical industries in a chain of manufacturing plants distributed along the belt from the New England States to the Gulf of Mexico. The location of this belt with reference to fuel, labor, climate, and the consuming and distributing centers could not be more favorable to rapid and economic development and profitable operation.

The Relation of Slow Driving to Fuel-Economy in Iron Blast-Furnace Practice.

BY JOHN B. MILES, PHILADELPHIA, PA.

(Chattanooga Meeting, October, 1908.)

THE present period of depression in the iron industry, with the resultant close approximation of the cost of production to the selling-price of pig-iron, should make the discussion of this subject at this time not only interesting but profitable. It is possible that it may result in the collection of data, which when added to those set forth in this paper will clearly indicate the influence of the rate of driving a blast-furnace upon its fuel-economy.

The question is not whether an excessive rate of driving (meaning a rate definitely in excess of the average rate in the United States) results in an increase in the fuel-consumption, but rather whether a rate considerably less than the average does not result in a marked decrease in the fuel-consumption.

The commercial and metallurgical sides of this problem must be kept separate. With a large margin between the cost of production and the selling-price of pig-iron, and a plant already in operation, it is frequently profitable to sacrifice a couple of hundred pounds of fuel per ton of pig-iron in order to secure the greater gross profits resulting from a larger tonnage.

When this margin decreases with a plant in operation, or when the determination of the size of a new furnace intended for a certain production confronts the management, a clear understanding of the effect of the rate of driving upon fuel-economy becomes necessary.

If it be granted that slow driving leads to low fuel-consumption, it should be borne in mind in the design of a new plant for the production of a certain tonnage that the adoption of an economical rate of driving increases only the size of the furnace proper, whereas, due to the decrease in quantity of blast required on account of the decrease in fuel burned, the stoves, the blowing-engines, the boilers and the piping may be de-

creased in capacity. The pressure of blast is decreased, which affects the boilers and blowing-engines and increases the surplus power if gas-engines are installed with the hope that a source of revenue will be found in the sale of power or its use in a nearby steel-plant.

It is hardly necessary to call attention to the commercial value of a decrease in the loss from flue-dirt as well as to a decrease in the difficulties due to its presence in the gas, with its adverse effect on stoves and boilers.

The rate of driving may be expressed in various ways, but for ease of calculation I have used, as an index, the consumption of fuel per sq. ft. of hearth-area per minute; thus paralleling the rate of driving of a boiler as expressed by the burning of so many pounds of coal per sq. ft. of grate-area. This naturally suggests the analogy between the relation of hearth-area to cubic capacity of the blast-furnace and the relation of grate-area to heating-surface in a boiler.

It would be still simpler to use as an index the piston-displacement of the blowing-engines per sq. ft. of hearth-area, but this introduces the efficiency of the blowing-tub, and the tightness of valves and piping, all of which are outside of the furnace, and renders this method of comparison absolutely unreliable, no matter how convenient it may be for use in directing the furnace-operation from day to day. Instead of the pounds of fuel, it would be still more accurate to use the weight of carbon burned per sq. ft. of hearth-area, but the analysis of the coke is not always obtainable, and furthermore, this refinement is not thought necessary.

The measure of economy in fuel-consumption is best expressed by the total burden of ore plus limestone carried per pound of fuel. In all comparisons, the silicon-content of the iron, the character of the ores used, and the quality of the fuel must be considered. No attempt has been made to reduce these total burdens in Table I. to a common basis whereby these factors would be introduced mathematically, and the comparative excellence of the furnace-practice in the various cases be finally set forth. It will therefore be necessary in comparing a high-silicon record with a low-silicon one, to make allowance for the difference in burden which ordinarily obtains in making pig-iron of different silicon-content.

It is not supposed that even the best of the records given in Table I. indicate the ultimate limit of burden-carrying capacity, and it is hoped that others may be given by members which will show even better work. No doubt some records may be produced showing excellent work with rapid driving, and doubt may result that the generalization connecting slow driving with fuel-economy has been proved. Nevertheless, it must be admitted that many of these records show remarkably good work of furnaces taking a quantity of air, in comparison with the size of the furnace, much less than many furnace-managers would consider feasible. In order that this effect may be more clearly seen, the quantity of air per minute has been calculated from the coke burned per minute, and has been introduced in Table I. under item *q*, on a constant nominal basis of 60 cu. ft. of air per pound of fuel.

Records Nos. 1 and 2 are of German furnaces using minette ore. The total burdens, given under item *o*, are remarkably high, while rates of driving, indicated by the fuel burned per sq. ft. of hearth per min. and given under item *p*, are the lowest in this collection of records.

Record No. 3 is of a furnace in the Middlesbrough district, England. The ordinary rate of driving in this district is about 2.5 lb. of fuel per sq. ft. of hearth-area per min., and the fuel-economy is excellent.

Record No. 4 is of Union furnace No. 1 of the Illinois Steel Co.[1] Although the total burden carried is not among the highest, yet remarkably low fuel-consumption was obtained on account of the richness of the ore. A comparison with the ores in use to-day is interesting.

Record No. 5 is of the furnace of the Penn Iron & Coal Co., taken from Arnold K. Reese's paper.[2]

Record No. 6 is of a charcoal-furnace.

Records Nos. 7 and 8 are of two furnaces with identical lines using Cornwall ore. These data are interesting because the difference in fuel-consumption shown continued over long periods consistently, the difference in the rate of driving being also constant, due to a difference in the blast-equipment.

Records Nos. 9 and 10 are of a furnace in Illinois. These

[1] *Trans.*, xx., 282 to 285 (1891).

[2] *Trans.*, xxvii., 477 to 485 (1897).

TABLE I.—*Data of Blast-Furnace Records.*

	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.	11.
a. Date	1904.	1904.	Oct., 1905.	Aug., 1890.	1895-1896.	1898.	Mch., 1899.	Mch., 1899.	Oct., 1906.	Dec., 1906.	Mch., 1908.
b. Period	Year.	Year.	28 days.	14 days.	181 days.	Year.	7 days.	7 days.	Month.	Month.	7 days.
c. Hearth diameter	14 ft. 5 in.	11 ft. 6 in.	11 ft. 0 in.	8 ft. 6 in.	10 ft. 6 in.	7 ft. 0 in.	10 ft. 0 in.	10 ft. 0 in.	15 ft. 6 in.	15 ft. 6 in.	13 ft. 6 in.
d. Bosh diameter	22 ft. 2 in.	22 ft. 2 in.	20 ft. 2 in.	14 ft. 10 in.	16 ft. 6 in.	12 ft. 0 in.	18 ft. 0 in.	18 ft. 0 in.	22 ft. 0 in.	22 ft. 0 in.	20 ft. 0 in.
e. Stock-line diameter	14 ft. 9 in.	16 ft. 2 in.	.....	9 ft. 4 in.	12 ft. 6 in.	8 ft. 0 in.	14 ft. 0 in.	14 ft. 0 in.	16 ft. 0 in.	16 ft. 0 in.	14 ft. 0 in.
f. Height	94 ft. 3 in.	82 ft. 3 in.	90 ft. 0 in.	72 ft. 4 in.	75 ft. 0 in.	60 ft. 0 in.	70 ft. 0 in.	70 ft. 0 in.	90 ft. 0 in.	90 ft. 0 in.	80 ft. 0 in.
g. Average daily product, tons	190	140	140	130	207	106	109	121	452	498	242
h. Yield of ore, per cent	.....	.....	41.5	63.5	59.0	52.0	.....	.....	51.0	52.6	52.2
i. Fuel per ton of iron, pounds	2,350	2,380	2,680	1,660	1,664	1,690	3,261	3,377	1,936	2,217	2,320
j. Ore per ton of iron, pounds	.....	.....	5,400	3,520	3,800	4,300	.....	.....	4,400	4,250	4,280
k. Limestone per ton of iron, pounds	.....	.....	1,830	606	771	.....	.....	.....	1,102	1,091	1,180
l. Ore plus limestone per ton of iron, pounds	7,320	7,620	7,230	4,126	4,571	.....	.....	.....	5,502	5,341	5,460
m. Ore per pound of fuel, pounds	.....	.....	2.01	2.12	2.20	2.55	.....	.....	2.27	1.92	1.84
n. Limestone per pound of fuel, pounds	.....	.....	0.68	0.36	0.46	.....	.....	.....	0.57	0.49	0.51
o. Ore plus limestone per pound of fuel, pounds	3.13	3.20	2.69	2.48	2.66	.....	.....	.....	2.84	2.41	2.35
p. Fuel per sq. ft. of hearth per min.	1.96	2.29	2.75	2.66	2.86	3.33	3.25	3.75	3.42	4.23	2.86
q. Air per min., from (*p*), calculated on basis of 60 cu. ft. per lb. of fuel	19,200	14,280	15,700	9,250	14,800	7,680	15,290	17,600	38,700	48,000	24,500
r. Average silicon in iron, per cent	Basic.	Basic.	1.00	1.6	1.23	1.0	.....	.....	1.27	1.10	2.50
s. Average sulphur in iron, per cent	.....	.....	.....	.....	.....	.....	.....	.....	0.044	0.049	.....
t. Temperature of blast, degrees Fahr.	.....	.....	.....	1,200	1,108	1,220	.....	.....	915	885	1,200

show the effect of an increased rate of driving, the blowing-equipment having been enlarged during the month of November.

Record No. 11 is of a furnace the hearth-diameter of which was made large in relation to the desired output when the furnace was designed, with the hope that the resulting slow driving would give a low fuel-consumption. The performance of this furnace from the beginning of its blast to the present time, indicates that no error was made in the adoption of this theory of design.

The usual rate of driving in the United States with a modern equipment is approximately 4.25 lb. of fuel per sq. ft. of hearth-area per min. With the data in hand, it is not possible to develop a rule for the most economical rate of driving with relation to fuel, for high- or low-silicon irons, but it appears that for either kind of iron, a rate less than 4 lb. is desirable, and it seems probable that a rate as low as 3 lb. may prove still more economical, so far as fuel is concerned.

If the records of the furnaces operated by the U. S. Steel Corporation were tabulated and published in some such form as that suggested by this paper, no doubt exceedingly valuable deductions could be made as to the influence of the rate of driving upon the fuel-economy. The knowledge thus obtained would be of great benefit to the blast-furnace industry.

Sulphur in Gaseous Fuels.

BY F. LOUIS GRAMMER, LEESBURG, VA.

(Chattanooga Meeting, October, 1908.)

THE difference between blast-furnace gas and ordinary producer-gas is chiefly that blast-furnace gas is higher in CO_2 and lower in hydrocarbons and hydrogen, as is shown in Table I.

TABLE I.—*Volume-Composition of Producer-Gas and Blast-Furnace Gas.*

	Producer-Gas. Per Cent. (Volume.)	Blast-Furnace Gas. Per Cent. (Volume.)
CO	22 to 33	25.5
CO_2	6 to 3	13.0
H	13 to 7	1.5
CH_4	3 to 1.5	0.0
N	54 to 60	60.0

If natural gas, water-gas, or by-product coke-oven gas be available, blast-furnace gas can be made to resemble very closely in calorific value the average producer-gas by merely mixing in one of these products, which are richer in hydrocarbons and hydrogen.

Blast-furnace gas, however, possesses a virtue which is sometimes absent in producer-gas, especially with coals very high in sulphur—namely, that blast-furnace gas is relatively free from sulphur. In some regions the sulphur in coal is an acute problem, and it is highly desirable to find uses for the mine-products which are specially high in sulphur-content.

An advantage would be gained if the coal were coked before charging into the gas-producers, but this adds to the expense.

Sulphur may be partly eliminated from coal by washing it, the success depending upon the form in which the sulphur occurs, and upon the efficiency of the washer, but it cannot always be removed to the extent desired at most American plants. It is not practicable to remove all the sulphur from producer-gas by washing it, though Mr. Reinhardt, of Germany,

refers to the use of bog iron-ore and wood-shavings as beneficial reagents for this purpose.[1] In making producer-gas from high-sulphur American coals the clinkering will be a nuisance, but it can be borne. The sulphur in producer-gas contaminates the steel in an open-hearth furnace, and to get a market for the sulphur-bearing coal and yet avoid the contamination of the steel, I suggest the following idea: For the open-hearth furnace, where sulphur is dreaded, use blast-furnace gas enriched with by-product coke-oven gas, water-gas, or natural gas. The deficiency in the regular supply of blast-furnace gas may be overcome by adding producer-gas, and using the mixture in the hot-blast stoves or under boilers, where the sulphur in producer-gas will do little harm.

The interchange of gas as suggested above is not an entirely new departure, since this practice has been followed for a long time in a pair of furnaces, one making ferromanganese, and the other, pig-iron. On account of the fact that the gas from the ferro furnace contains dust which would slag the brick in the hot-blast stoves, it is used only under the boilers of both furnaces, while the gas from the other furnace is used for the stoves of both furnaces.

Blast-furnace gas need not be washed to the point of purity desired for use in blowing-engines—namely, 0.01 g. of dust or less per cu. m., but it can be reduced to 3 or 4 g. per cu. m. by dry purifiers, provided very fine ores are not used.

If the open-hearth steel-furnace be near a blast-furnace, and if enriching gases are available, the transfer suggested above might help in many instances, since the cost of installation is slight.

Since writing the above notes, I have heard that at a plant in Scotland fuel high in volatile matter was used in the blast-furnace, and the resultant blast-furnace gas was used in the open-hearth steel-furnace.

The desire of American rail manufacturers to make large outputs and a low percentage of seconds at the mill has tended to lower the sulphur-limit in the metal furnished to the mill.

In the western United States the question of sulphur-content has not been an acute one; still, some ores and all cokes contain

[1] *Trans.*, xxxvii., 685 (1907).

more sulphur than is desired. In Europe, however, more attention has been paid to this question, and several special processes have been devised to reduce the sulphur-content of molten metal, and also to remove the sulphur completely from several iron-ores before charging the furnace.

While Mr. York[2] in describing his universal mill does not claim that it will handle metal high in sulphur with a less proportion of seconds, this result could reasonably be expected, and his mill should be specially popular in those sections in which the ore and fuel are not as free from sulphur as are the Lake ores and Pocahontas cokes. This feature applies equally well to the mills of Wenstrom, Sack, or Grey.

Although efforts have been made to get foreign rail-buyers to reduce the limit of phosphorus, but little attention has been given to the sulphur-limit, the foreign buyers in some instances allowing 0.080 per cent. of sulphur in the rail.

Greater flexibility in the sulphur-specifications in steel would help many sections of the United States, for keeping up silicon and keeping down sulphur increases the cost and diminishes the blast-furnace product; possibly the new mills will permit this desired flexibility.

An Unusual Blast-Furnace Product; and Nickel in Some Virginia Iron-Ores.

BY FRANK FIRMSTONE, EASTON, PA.

(Chattanooga Meeting, October, 1908.)

THE material described below was made at Furnace No. 2, Longdale, Va., Oct. 28, 1907, during a sudden derangement of working, in the course of which the furnace became entirely bridged over and for some time no solid material whatever descended into the hearth. This caused the hearth to become so completely emptied that, at the tuyere-level, a pricker put in at one tuyere could be seen by looking in at the opposite one. The tap-hole was opened and the iron got out with some difficulty. It ran sluggishly, throwing many sparks, but was not

[2] *Trans.*, xxxvii., 859 (1907).

too thick to run into the chill-molds. Naturally, the product was assumed to be white iron, which is almost always produced under similar circumstances; but on removal from the molds it proved to be very strong and tough, although full of blow-holes. Most of these blow-holes had bright, unoxidized walls. The iron could be drilled by using a "special steel" drill, the drillings curling up similarly to those of steel. An analysis by P. W. Shimer of this unusual product gave: Si, 0.019; P, 0.228; S, 0.39; Mn, 0.042; C, 2.366; and graphite, 0.000 per cent.

A cast of iron made just before the disturbance contained P, 0.98, and Mn, 1 per cent.

The peculiar product seems to have been decarburized in the hearth and, at the same time, to have lost much phosphorus and manganese, either by the action of the blast in the empty hearth, or, more probably, by contact with the cinder rich in iron which was formed at the time. Unfortunately, no specimen of the cinder was preserved, so its composition cannot be given.

It seems very remarkable that, with the furnace in bad condition, the temperature should be high enough to keep such "steel" sufficiently fluid to permit of its running into the chill-molds. The circumstances must have been quite exceptional, for, under apparently like conditions, I have never seen or heard of anything but ordinary white iron running out at the tap-hole.

The present occurrence is of further interest in connection with experiments of Prof. Bernhard Osann on the composition of salamanders[1] (*Bodensauen*).

As always happens in the case of furnaces using the so-called Oriskany ores of Virginia, there was much evidence of the presence of zinc in the hearth while the trouble lasted. At my suggestion, Mr. Shimer made a careful examination for zinc in the "steel," but found none, which is in accordance with previous experience.[2] A special point of interest, however, was the unexpected discovery of an appreciable amount of nickel and cobalt, chiefly nickel, which amounted to 0.184

[1] *Stahl und Eisen*, vol. xxvii., No. 42, pp. 1491 to 1496, and No. 43, pp. 1529 to 1536 (Oct. 16 and 23, 1907).

[2] Percy, *Iron and Steel*, pp. 153–154 (1864).

per cent. An examination of other specimens of pig-iron from Longdale and from two other furnaces in the vicinity, all using Oriskany ores but mined at widely-separated points, gave the following results:

		Nickel and Cobalt. Per Cent.
Longdale, ...	June 18, 1881	0.100
Longdale, ...	Jan. 21, 1908	0.107
Iron Gate, ...	 1908	0.155
Princess, ...	 1908	0.045

The pig-iron from the Iron Gate furnace was obtained through the courtesy of the manager, William W. Taylor, and that from the Princess furnace from J. E. Johnson, Jr.

It is probable that a small amount of nickel is widely distributed in the Oriskany ores, and that it will be met with generally, as is the case with the small but very troublesome percentage of zinc which seems never to be absent.

Prof. E. D. Campbell[3] reports the presence of a small quantity of nickel and cobalt in iron from a Virginia furnace, but gives no indication of the localities from which the ore-supply was obtained.

I was not present at Longdale when the "steel" above described was made, but Capt. J. E. Johnson, the general manager at Longdale, has kindly given me the facts and has read over what I have written above. To my great regret he has, from pressure of business, declined to describe the occurrence himself.

[3] *Journal of the Iron and Steel Institute*, vol. lxviii., p. 371 (No. II., 1905).

Sulphur Dioxide as an Agent in Fighting Mine-Fires.

BY WALTER O. SNELLING, WASHINGTON, D. C.

(New York Meeting, February, 1908.)

In combating mine-fires the use of carbon dioxide as a means of producing an atmosphere in which combustion cannot be sustained, has been many times suggested and frequently tried, generally with a fair degree of success. The carbon dioxide is produced by acting on limestone with dilute sulphuric acid, large wooden boxes being used to hold the limestone, and arranged in such a manner that one box may be cleaned and refilled with blocks of limestone while another is being used in generating the carbon dioxide The gas is led by means of pipes of large diameter to the location of the fire, tight brattices having been previously put up as close to the fire as possible on every side, so as to limit the space to be filled with the gas to the smallest possible volume. The cost of producing carbon dioxide has been, however, a decided draw-back to the use of this method, and the danger of producing carbon monoxide by the reduction of the carbon dioxide by heated carbon, and consequently bringing about an explosion, has also tended to prevent the application of this method.

The use of sulphur dioxide in combating mine-fires has not, I believe, been previously suggested or tried, and yet the method presents such decided advantages over the use of carbon dioxide that it seems proper to suggest the advisability of considering it as a cheap, convenient, and safe means of fighting stubborn mine-fires.

1. *Cost, Preparation, etc.*—One ton of brimstone, costing from $20 to $30, will produce about 25,000 cu. ft. of sulphur dioxide gas, and allowing for all sources of loss it is probable that sulphur dioxide gas can be produced for about $1,300 per 1,000,000 cu. ft. For carbon dioxide gas, produced by the action of sulphuric acid on limestone, a good estimate would be $4,000 per 1,000,000 cu. ft., provided that sulphuric acid

and limestone could be obtained at a small cost. Even under these favorable conditions it will be seen that carbon dioxide is about three times as costly as sulphur dioxide, and under ordinary conditions the advantages on the side of sulphur dioxide would be much more marked.

2. *Efficiency.*—Sulphur dioxide is more efficient than carbon dioxide in the putting out of fire. Neither coal nor any other combustible material can possibly burn in an atmosphere containing any considerable quantity of sulphur dioxide. Sulphur dioxide is very heavy, being almost twice as heavy as carbon dioxide (1 cu. ft. weighing 81 g., while 1 cu. ft. of carbon dioxide weighs 56 g.), and this heavier weight of the sulphur dioxide is also an element which leads to increased efficiency, since the readiness with which the air present in the interstices of a pile of burning coal will be displaced by any inert gas is dependent upon the density of the gas, and as sulphur dioxide is twice as heavy as carbon dioxide, it will have about twice the efficiency of the lighter gas in displacing the air from piles of burning material, and thus removing the oxygen available for continuing the combustion. Another important effect of the greater density of sulphur dioxide is that its cooling effect upon a bed of incandescent coal is greater, volume for volume, than would be the case with a less dense gas.

3. *Safety.*—None of the dangers incident to the use of carbon dioxide are present with sulphur dioxide. No explosive lower oxides are produced or can be produced by the reduction of sulphur dioxide. Still more important, the danger always present when using carbon dioxide, of men getting into the gas without knowing of its presence, and suffocating, is absent when sulphur dioxide is used, since its strong odor gives instant warning when but a small fraction of 1 per cent. of the gas is present in the air.

4. *Other Advantages.*—Any leakage of carbon dioxide from a mine can only with great difficulty be detected. Frequently, in places where this method has been applied, it is probable that millions of cubic feet of the gas have escaped through unknown fissures in the rock. Any leakage of sulphur dioxide would quickly make itself known, and the openings could accordingly be closed before more than a small portion of the gas had escaped.

The keeping of oxygen-helmets as emergency apparatus at coal- and other mines is believed to be increasing in the United States, and it is to be most strongly advocated. When such oxygen apparatus is at hand it will nearly always be possible to send men from time to time to see if the mine-fire is extinguished before turning the ventilation into that portion of the mine. Most forms of rescue-masks can thus be worn in an atmosphere of sulphur dioxide, so it is possible to know definitely when the fire is extinguished, and when it is safe to turn in the fresh air. But when the sulphur dioxide method has not been used, the large amount of carbon monoxide present is likely to be a source of danger to the person wearing the helmet, since it is an interesting fact that some of the simpler patterns of rescue-helmets which may be worn with safety in an atmosphere of sulphur dioxide are found to be but slight protection when worn in an atmosphere containing much carbon monoxide. The reason for this is not clear, but it is most probably due to the greater permeability to carbon monoxide of the leather or other fabric of which the rescue-mask is made, the lighter carbon monoxide passing through the fabric with greater readiness than most other gases.

5. *Conclusion.*—The extremely simple manner in which sulphur dioxide can be produced, the cheapness of the method, and the efficiency of the gas in putting out fires, seem to make this a desirable means of combating mine-fires. The safety of the method, its convenience, and its utility in showing fissures which allow escape of gas and ingress of air, are also strong points in its favor. The following comparison of the physical properties and cost of producing sulphur dioxide and carbon dioxide will be of interest.

	Sulphur Dioxide.	Carbon Dioxide.
Molecular mass, . . .	64.	44.
Mass of 1 liter, . . .	2.87 g.	1.97 g.
1 cu. ft. weighs, . . .	81.13 g.	55.78 g.
1,000 cu. ft. weighs, . .	178.48 lb.	122.71 lb.
1,000 cu. ft. requires, .	89.24 lb. sulphur.	300 lb. H_2SO_4. 296 lb. $CaCO_3$.
Cost materials per 1,000 cu. ft.,	$1.35	$4.00
Cost of 1,000,000 cu. ft., .	$1,350.00	$4,000.00

In effectiveness, safety, and other considerations the advantages are also in favor of the sulphur dioxide.

The Chinese on the Rand.

BY T. LANE CARTER, JOHANNESBURG, SOUTH AFRICA.

(New York Meeting, February, 1908.)

BEFORE describing the experience with the Chinese on the Rand and the work they have accomplished, it will be necessary, first, to give a brief account of labor-conditions in the Transvaal since the war, and then to show how the necessity of importation of labor arose.

As is well known, the Transvaal mines are run on the dual system of labor—that is, the rough work is done by colored laborers under the superintendence of white men. From the start of mining operations in South Africa there has been the cry of shortness of native labor, due in great measure to the failure of Kaffirs to work continuously in the mines. While he works, the Kaffir is a satisfactory laborer, but his period of service is short, averaging about nine months.

People unacquainted with the Kaffir have maintained that, by offering a higher wage and better accommodation, better results would be obtained from the native workers, but an increased wage simply aggravates the trouble. The Kaffir's wants are few. While at work he spends little of his money, and before long has sufficient capital with which to buy wives and become a gentleman of leisure. Doubling the wage of the Kaffir means that he acquires a competency in half the time, so that his period of service is correspondingly reduced.

In 1899, before the war, there was always a cry about the scarcity of Kaffir labor. The highest number of natives at work was in July and August, 1899, when nearly 100,000 were employed. At the outbreak of hostilities the Kaffirs were scattered far and near, many doing well during the war by working for the military authorities.

On the resumption of mining operations on the Rand a serious mistake was made. Based on past experience, it was argued that if the whole industry decided to start anew with

half the wages given the Kaffirs before the war, then the length of service of the natives might be doubled. It was agreed, therefore, to cut the wages of the Kaffirs in half, before any mines had re-started or any Kaffirs had come forward looking for work. But when the black men heard of the new rates they decided to keep away from the mines. In spite of every effort made to obtain labor, very few came to the Rand. Although the wage was gradually raised to that prevailing before the war, the Kaffirs did not come forward. Up to the time of the introduction of the Chinese, the largest number of Kaffirs at work on the Rand since the war was about 64,000, in June, 1903.

Something had to be done; the mines could not run much more than half of the time. It was impossible to push development-work ahead, and the ore-reserves grew less and less. To augment the labor-supply, the work ordinarily done by Kaffirs was offered to hundreds of white men who remained in the country after the war. Many mines employed white men at wages varying from $1.20 to $1.80 per shift, but all experienced men soon came to the conclusion that to make up the shortage of Kaffir labor by the use of white men was out of the question. Every one saw that, without a greatly increased labor-supply, there could be no expansion on the Rand. A commission appointed by Lord Milner reported that South Africa could not supply sufficient Kaffir labor for all requirements. There seemed no alternative but to import labor for the mines.

At first the idea of Chinese was repellant to every one, but as labor became scarcer the majority began to look upon the yellow man as the only solution of the difficulty. A small minority bitterly opposed the introduction of Chinese at any price, and have never ceased in their war against the Celestial.

There was some talk of getting labor for the mines from India, but the objections to Indian coolies are that they are inferior to Chinese as workmen, and that, being British subjects, it was impossible to get as severe an indenture for Indians as for Chinese immigrants. One colony in South Africa had already used Indian labor to develop its industries, and the Rand had no desire to repeat the experience of Natal, where the Indian has been so successful in ousting the white man.

The Chinese Labour Importation Ordinance, under which coolies are imported into the Transvaal for the mines, is a unique document of its kind. The skilled white man is protected against the encroachments of the Chinese more completely under this ordinance than in any other country in the world. There is no reason, as far as the law is concerned, why every white man on the mines should not be replaced by a Kaffir. In the past, the lack of capacity of the Kaffir has been the white man's only protection.

But it is quite different with the Chinese coolie. He is brought to the Rand for a definite purpose—that of a rough laborer—and he is forbidden by law to do skilled work. For every offense, the person or persons who employ him otherwise than as an unskilled laborer in the exploitation of minerals, in the Witwatersrand district, are liable to a fine of $2,425, and, in default of payment, to imprisonment for a period not exceeding two years. The ordinance is very explicit in its definition of the work that the coolies are allowed to do.

If, at any time, the white men find the ordinance is being violated and the Chinese are doing skilled work, they can lay a charge against the manager of the mine for breaking the law. For instance, I was summoned before a magistrate quite recently at the instigation of a labor Member of Parliament on the charge of allowing a Chinaman to perform skilled work on a Cornish pump. From the evidence, it was clearly proved that I had placed a white man in charge of the pump, but that he had gone off and allowed the Chinaman to do the skilled work. The magistrate held that the ordinance was not broken and dismissed the charge. This instance shows the protection that the ordinance gives to the white artisan.

Of course, the white men have occasionally endeavored to read meanings into the ordinance which never existed. Not a word is said as to the number of coolies a white man is allowed to superintend. As the Chinese became more efficient, the management of mines requested the white men to "boss-up" more coolies than formerly. The men refused, however, and on this as one of the issues the only great strike ever known on the Rand was fought out, the men being woefully beaten in the struggle. At present, the ratio varies on the different mines, and is 1 white man to from 8.5 to 12.5 coolies.

The general administration of the ordinance is in the hands of a superintendent, who is given a great deal of authority. Under him, in the different districts, are inspectors. At first there was considerable friction between the mines and the Foreign Labour Department over the control of the coolies, the mine-managers maintaining that more authority should be given them in punishing offenses; but this friction has passed away, and the present system of dealing with Chinese offenders is, on the whole, satisfactory to all concerned.

The inspectors, of whom there are eight on the Rand, have the power of magistrates, and visit each mine once or twice a week to try the offenses committed by Chinese. On each mine, near the compound, there is a jail, in which to lodge the culprits temporarily until they are tried by the inspector. Unfortunately, the inspectors have not the authority to order corporal punishment save in rare instances, when the case must be first submitted to the Attorney-General. There is no punishment a coolie dislikes more than corporal punishment: and as he has been accustomed to severe discipline under Chinese law, the penalties for wrong-doing under English law appear to him very mild. The wonder is that the Chinaman has proved so law-abiding during his term of service. The inspectors punish the coolies by imposing fines and sentencing them to terms of imprisonment.

Under the Labour Importation Ordinance no Chinese coolie may enter the colony except under contract of service. Any person bringing in a coolie, unless he has obtained a license to do so, is liable to a fine of $485 for every laborer introduced. All Chinamen introduced under the ordinance must be employed in the Witwatersrand district. The laborers must be returned to China without delay at the end of their three-year contract. Provision is made, however, for a renewal of contract for a period or periods not exceeding in all three years.

Under no condition are coolies allowed to trade, or to acquire, lease, or hold land. They are not allowed to roam around the country without a passport.

The ordinance provides for the immediate return to China of undesirable characters and criminals, as well as laborers who have become incapacitated for work through sickness or accidents.

Although very few Chinamen have availed themselves of the privilege, the ordinance provides for the introduction of the wives of laborers, and their children less than ten years of age, at the expense of the importer. Once here, however, the laborer is bound to maintain his family at his own expense.

Any laborer may terminate his contract of service at any time, without assigning any reason, on tendering to his importer the expenses incurred in introducing him (and his wife and children, if any) into the Transvaal, together with a sum sufficient to defray the expenditure necessary in returning them to China.

Every mine is required to provide laborers, and their wives and children, with medicine and medical attendance free of charge, and any employer who neglects to do so is liable to a fine of $242.50 for each and every offense.

The employer is bound to give work to every laborer for six days in every week, except on Sundays, Christmas day, and Good Friday, and on the days of the four Chinese festivals.

Save for one lot of southern Chinese, all the coolies that have come from China are from the northern part of the country. At the beginning of the introduction of Chinese labor there was such a rush to get the coolies to the Rand that a large number of undesirables, criminals, and beggars came to the Transvaal. The last batches to arrive were a far better class of Chinamen.

Many difficulties had to be met in getting the coolies started on the mines. In the first place, such monstrous tales had been spread along the Rand about the Celestials—of their treachery, their vice, their cruelty, that the white artisans were afraid to work with them. In consequence, when the Chinese arrived, the best artisans went to the mines run entirely with Kaffirs, leaving a very poor class of men to superintend the coolies. Gradually the feeling of fear of the coolies passed away, and to-day many artisans, especially rock-drill miners, would rather work with Chinese than with Kaffirs.

Then the rumor was circulated that the Chinamen were so frugal that they would hoard all their money and take it out of the country, a rumor which of course put the commercial community against the Chinese. Instead of proving misers, the coolies are veritable spendthrifts. The store-keepers near

the different compounds have found the Chinese far better customers than the Kaffirs. Their tastes are more refined than those of the natives, and they purchase a more expensive class of goods. So lucrative are these Chinese stores that the demand for stands near the compounds is very great. All the stores are run by independent store-keepers, the company charging only a nominal rent, where it owns the land and store.

The Chinaman's tastes are peculiar. His special fancy is for clocks and watches. In most cases he has abandoned his Eastern costume and now dresses in European clothing. Unlike the Kaffir, he always wears boots in the mine, so that the army of coolies on the Rand make a big business for the boot-trade. The coolie spends his money freely on food-stuffs, and has taken a great fancy to bottled lemonade, especially if it be pink or red. He is fond of making fancy dishes of his own, and most of the mines allow the coolies to use a small kitchen to prepare their dainties.

The Chinamen are housed in large quarters, called compounds. The rough plan of the French Rand compound, shown in Fig. 1, gives an idea of the Chinese barracks. Hygienic regulations are enforced, and the number of Chinamen in a room is limited by the fact that for each inmate there must be 200 cu. ft. of air-space. In the large rooms (30 by 30 ft.) as many as 50 coolies are sometimes placed, there being a double row of bunks. Ventilation is amply provided for, and in the winter time there is a stove for each room. Disinfectants are freely used. On the whole, the compounds are kept clean, and so far there has been no epidemic, such as small-pox, among the coolies.

Two systems of feeding the laborers are in vogue on the Rand. In one there is a large dining-house where all the coolies sit together at their meals. In the other the coolies are divided into messes of ten, and rations for the mess are served out to one of the members twice a day. This man takes the food to the sleeping-rooms, where it is eaten. On going to work in the morning, each coolie is given a loaf of bread and a bottle of tea for his mid-day meal. About 18 coolies are kept constantly at work in the compound cleaning up. The rooms are flushed out frequently with water.

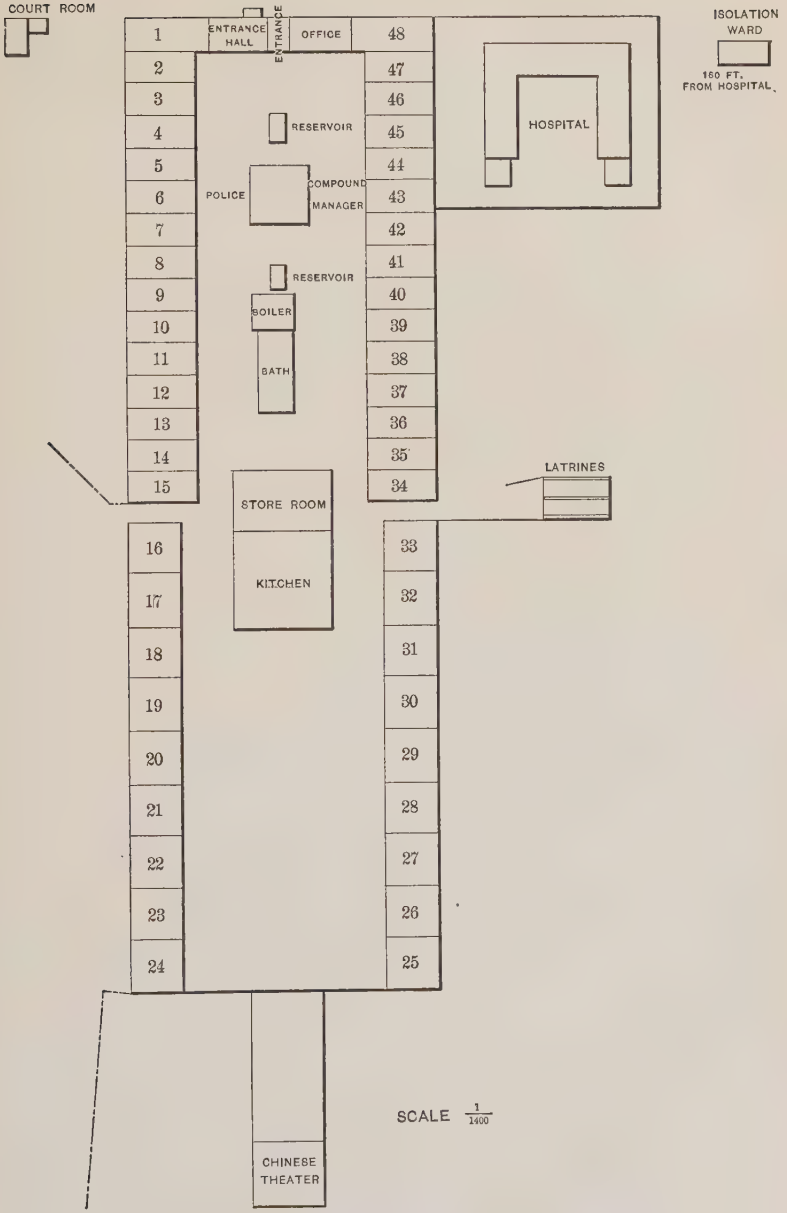


FIG. 1.—THE FRENCH RAND COMPOUND.

The ordinance allows the following rations per man per day: 1.5 lb. of rice; 0.5 lb. of dried or fresh fish or meat; 0.5 lb. of vegetables; 0.5 oz. of tea; 0.5 oz. of nut oil; salt. The

coolies are provided free of any charge with housing, water, fuel, and medical attendance.

By experience, the rations allowed are found rather excessive, and to prevent waste are reduced in some cases, but it is not the policy of the mines to save on food. The laborers can have as much as they please. Very often, as a change, they buy bushels of fruit brought around the compounds for sale by Boer farmers.

The vegetables given daily to the coolies are generally supplied on contract by farmers. Some companies have attempted to grow their own vegetables, but experience has shown that it is cheaper to buy them on contract. The price paid for fresh vegetables at the compounds averages all the year $2.05 per 100 lb., a figure which compares favorably with costs in most other places.

The compounds are under the direct supervision of a controller, or "compound manager," and his assistant. With the first batch of coolies there arrived a number of white men conversant with the Chinese language. Some of these men got positions as controllers and assistant controllers. In many cases the men who had acted as Kaffir controllers were placed in charge of the Chinese. On account of their ignorance of the coolies and their language, these men were rather lost for a while, but they rapidly learned the dialect, and to-day some of the most successful controllers on the Rand are men who a few years ago knew nothing about the Chinese.

Each controller has under him a number of Chinese police, at the head of whom is a Chinese sergeant. Fig. 2 shows a number of Chinese police at the French Rand mine, and Figs. 3 and 4 represent characteristic groups of the coolies; these photographs were taken in one of the compounds. One policeman is allowed for about every 80 laborers. The duties of the police are to keep order, to turn the coolies out to work, etc. On several occasions when riots have broken out in the compounds they have had a very hard time. Unless the police are carefully watched, grave abuses arise in the compounds. Experience has shown that a Chinaman is a dangerous man to whom to give unlimited authority. The "squeeze" system seems to be so ingrained into every Chinaman that if he is given power he abuses it, and soon makes a kind of mandarin

FIG. 2.—THE CHINESE POLICE ON THE FRENCH RAND MINE.

FIG. 3.—COOLIES IN A CORNER OF THE COMPOUND.

FIG. 4.—A GROUP IN THE COMPOUND, ON THE FRENCH RAND MINE.

of himself, extorting money from his subjects. Before the coolies can gamble or smoke opium, the Chinese police have to be "squared."

So accustomed are the coolies to the "squeeze" system that they look upon it as human nature, considering it a trait of all nations. When I tried to introduce a slight innovation, by which the coolies were required to do a little more work for the same money, the Chinese in talking it over among themselves declared I was getting my "squeeze" out of them by requiring harder work, and that all money saved went to my pocket.

After the arrival of the Chinese there was for a time a large number of desertions from the mines every week. In many instances these deserters attacked out-of-the-way houses, with the hope of gaining a lot by plundering. Sometimes one or two of the inmates were murdered. It is creditable how these desertions have been stopped.

There are many causes for the cessation of desertions. In the first place, the finger-impression of every coolie on the mines is filed at the Foreign Labour Department. Whenever a stray Chinaman was found roaming about the country without the authorized permit he was readily identified and, after undergoing several months' imprisonment for unlawful absence, he was sent back to his mine. A careful check-system was followed at each mine, and as soon as it was found that a coolie was absent, the police all over the Transvaal were notified.

By degrees most of the bad characters were found out, and after several months' imprisonment they were sent out of the country. Even men of suspicious character, against whom nothing could be proven definitely, were in many cases sent back to China.

Another reason for the decreased desertions is to be found in the fact that many of the coolies were killed or wounded by the occupants of farms or stores, when the Chinamen made their raids at night.

The Chinamen is a sensible individual, and he came to the conclusion after a while that running away from the mines, either to escape paying his gambling or other debts, or to go on marauding-expeditions, did not pay, so he decided to remain in the compounds and not leave the mine without a legal permit.

These permits are readily granted on Sundays to the coolies by the controller, and hundreds of the coolies ride in the trains from one station to the other to visit their friends. So effective have the measures to prevent desertions proved, that the number has fallen from a maximum of 689 per week to 75 at the present time. When a Chinaman wishes to escape his creditors in the compound now, he does not desert as formerly, but remains underground altogether for several weeks at a time.

The large number of desertions of the Chinese and the outrages they committed were made a great deal of by the anti-Chinese party. The crimes committed by the coolies were deplorable. Unfortunately, there has lately been a recrudescence of Chinese crime during the past week, after a period of many months when there was scarcely a serious outbreak. As their contracts fall due, many of the coolies, who have lost all their money by gambling, seem to think that by plundering they can secure sufficient money to take them back to China.

It must be remembered, however, that with a total of 53,828 Chinese, the number of serious offenses has not been higher than for the same number of men of any other nationality living under the same conditions. On many occasions the same offense committed by a Kaffir or a white man would have passed almost unnoticed, but because the offender was an imported coolie, the press and the public gave more prominence to the crime than was warranted. On the whole, the indentured laborers have conducted themselves fairly well, and compare favorably with other peoples.

It was stated when the Chinese first arrived on the Rand that their influence on the Kaffir would be rather demoralizing, that the yellow man would teach the aborigines opium-smoking and other vicious habits. On the whole, however, the Kaffir has had the more demoralizing effect on the Chinaman. I have not yet known of a case where the coolie has taught a Kaffir the use of opium, but any number of cases of Kaffirs instructing the Chinaman in the use of alcohol have come to my notice. As to other vice, the coolie is not one whit worse than the Kaffir.

During the recent elections in the Transvaal, one of the most persistent arguments urged against Chinese labor was the

fact that Central Africa was teeming with natives, and that the Rand could be kept going with natives from these parts. This contention was completely "squashed" by Drs. Irvine, Napier and Macaulay in a published article[1] which shows that the awful death-rate among Central African natives on the Rand is not due to neglect; and that these men live under exactly the same conditions as other Kaffirs, who show low death-rates. The death-rate on the Rand of natives from different parts of Africa is as follows:

(A) Areas which May be Classed as Temperate.

	Mortality per 1,000.
Orange River Colony,	13.9
Natal and Zululand,	14.9
Cape Colony,	19.8

(B) Areas which May be Called Sub-Tropical.

(The death-rate for the Transvaal is greatly raised by the inclusion of natives from the northern Transvaal districts.)

Southern Portuguese East Coast districts (south of lat. 22°),	35.0
Transvaal,	36.0

(C) Areas which Are Tropical and Highly Malarial.

Northern Portuguese East Coast districts, viz.:	
Mozambique,	65.8
Quilimane,	75.6
Rhodesia,	79.8
Damaraland,	113.8
British Central Africa,	166.3

The normal death-rate of 166.3 per 1,000 is seldom met anywhere else in the world. Experience proves that although this high death-rate of Central African natives can be reduced somewhat, the natives from this part of Africa cannot stand continuous work on the Rand. It is therefore absurd for the anti-Chinese advocates to say that the coolies can be replaced by natives from Central Africa.

Kaffirs from the malarial tropical districts cannot stand the climate of the high veldt.

The comparative mortality for whites, Kaffirs, and Chinese on the Witwatersrand districts is, per 1,000:

[1] *The Star*, Nov. 10, 1906.

Period.	Natives.	Chinese.	Whites.
July, 1905, to June, 1906, . .	43.335	19.644	19.358
July, 1906, to June, 1907, . .	33.807	17.045	18.464

The improved mortality of natives is due to the fact that greater care is now taken in the recruiting of natives. The Kaffirs must pass a strict medical examination before they are allowed to proceed to the mines. All weaklings and "medically unfits" are sent back to their kraals.

In one sense the influence of the Chinese on the Kaffirs has been of inestimable benefit. Ever since mining operations commenced in South Africa the black men have had a monopoly as rough laborers. They are the most independent workmen in the world, and if the terms do not suit them they remain at home, as they did after the war. Knowing that there is always a strong demand for their services, they come to work when it pleases them. With the advent of Chinese there appeared for the first time competitors. To us who have been in South Africa for some years, and have had a continual worry to get Kaffirs to work, it was a real pleasure to see numbers of black men come voluntarily to the mines asking for work. Generally we have to send for them.

That the presence of 53,828 Chinese on the Rand has had a most stimulating effect on the Kaffirs there can be no doubt. While the coolies are here, the aborigines realize for the first time that they have not a monopoly of the unskilled labor market.

For political reasons much has been made of the cry of Chinese vice by men both in England and in the Transvaal, and the way reports have been written on this subject is nothing short of scandalous. The favorite method is to send an imaginative journalist out from England with a commission to damn the Chinese-labor question on the Rand at any price. This gentleman spends his time at one of the leading hotels, interviews anti-Chinese advocates, picks out of the daily papers cases illustrating the depravity of the Chinese, and then writes his confidential report, making the most unjust and sweeping accusations. It never seems to occur to him that he should visit the mines to interview the managers and controllers, and gather information at first hand.

Visitors are usually surprised at the excellent hospital-ac-

commodations. The hospital is a separate building, close to the compound, and in some cases is superior to the hospital for whites on the same mine. Quite a number of Chinese doctors, with European training, came to the Rand with the first lot of Chinese, but these men were not successful, and most of them have gone back to China. The hospital is in charge of a resident physician, who has under him a qualified medical assistant. There are also several coolies who act as nurses, dressers, etc., and who are excellent at this work. The hospitals are divided into two wards, for accidents and for diseases. A large operating-room and a dispensary are part of the building.

Just here it may be appropriate to mention the vital statistics of the Chinese, and to show their superiority to the Kaffirs in standing the climate. In fact, they are almost as hardy as the whites. While the statistics show in favor of the coolies, it must not be forgotten that the Chinese are picked men. The comparison with the Kaffir is fairer.

It has already been shown how well the climate of the Rand agrees with the health of the Chinamen. At first there was a scare of the disease beri-beri, the death-rate for a time being 4.236 per 1,000 from this cause. The disease has been fought so successfully that now beri-beri is almost extinct, the death-rate at present being only 0.169 per 1,000. For the past four months the total death-rate of the Chinamen on the Rand was 16.735 per 1,000 per annum.

There was also some apprehension that the disease ankylostomiasis (miner's worm) would break out among the coolies, but so far it has not given much trouble.

Where the Kaffir seems weakest, the Chinaman makes the best showing—namely, in pulmonary complaints. During the winter months, June, July, and August, the death-rate among Kaffirs from pneumonia is very high, whereas the coolies suffer little from this disease.

It is on account of his splendid vitality that the coolie is such an ideal laborer for the Rand. There can be no question of the superiority of the Chinaman over the Kaffir as far as vitality is concerned. When it is remembered that, in some months of the year, the natives from Central Africa at work on the Rand show the awful death-rate of 166 per 1,000 per annum, it seems inhuman to bring the black man from the malarial dis-

tricts of Central Africa to this high altitude when Chinamen can be engaged to work the mines.

The coolies suffer most from bronchial troubles. The number of deaths from suicide among the coolies is very considerable. In fact, the tendency of Chinese to commit suicide is a distinct race-trait. Their favorite method is to drink a solution of opium. If detected in time, their lives may be saved. During the past three years fully 20 Chinamen have been rescued from death by opium-poisoning at the French Rand mine.

The Chinese vice of opium-smoking has given a great deal of trouble, but, on account of the strong measures taken by the government, the curse has been kept within limits. In very bad cases the coolie can obtain permission to smoke a small amount of opium. Promiscuous smoking, or being in possession of opium, is unlawful, and is punishable by fine and imprisonment. There is a considerable illicit trade carried on, however, in spite of fine and imprisonment.

The breaking-in of the coolies to mine-work was a trying ordeal. Not only were the Chinamen raw at such work, but the managers and men knew practically nothing of the coolie or his language. Such monstrous tales were circulated along the Rand concerning the "heathen Chinee" before their arrival, that no miner would work with them if it was possible to get work at mines where Kaffirs were employed. Consequently, the managers of Chinese mines had a very hard task; they had to break in hundreds of green Chinamen with very inferior white men to help them. But the artisan's opinion of the Chinamen has changed, and to-day a large number of the white men prefer working with Chinamen.

Every one was handicapped by his ignorance of the coolies' language. Few of the imported laborers had ever seen a mine before. In China they were farmers, beggars, and criminals. Their new work demanded a lot of explanation, and the white man in charge, after a vain endeavor to explain the work, lost his temper and a row ensued. Soon the white man learned the Chinese "cuss words," and, as the Chinamen use these expressions often, the white man thought the coolie was swearing at him, and blows were passed. To-day the white man and the Chinaman can make themselves understood by a language of their own, and the friction between them has passed away.

Another serious difficulty at first was the danger to the coolies from accidents. Being quite new to mining-work, they possessed the faculty, common to raw recruits, of being in the wrong place at the wrong time. For a few weeks after their arrival the number of coolies killed or wounded by fall of rock, blasting, and machinery was very great.

The majority of the accidents were caused by the coolies drilling into old holes containing a stick or so of gelatine, thereby causing an explosion. The mining-regulations require the white man in charge to plug up all old holes, but often the coolies, in spite of all warnings, would withdraw the plug and start drilling in the old hole. Formerly, the death-rate per thousand of the Chinamen from accidents was 5.053. In spite of their experience, the death-rate from accidents has increased to 5.754. Like the Kaffir, the Chinaman is very careless underground.

Feeding coolies costs much more than feeding Kaffirs. The cost for the Kaffir per shift was $0.0801, made up as follows:

Meal,	$0.030	Sundry,	$0.011
Meat,	0.036		
Vegetables,	0.003	Total,	$0.0801
Other food,	0.0001	(Practically 8 cents per shift.)	

Food-stuffs are bought wholesale, the prices in 1907 along the Rand being:

Rice, per bag (224 lb.),	$3.30	Tea, per case (50 lb.),	$8.30
Mealie meal, per bag (200 lb.),	1.95	Vegetables, per 100 lb.,	2.06
Meat, per lb.,	0.08	Salt, per bag (180 lb.),	1.93
Bread, per 100 lb.,	3.87		

The food is plain but nourishing, and compares favorably with the rations of miners in Europe. The cost per man per day shows what can be done, even in this expensive country, in the way of feeding men cheaply on a wholesale scale.

As to the feeding of the Chinamen, a greater variety of food is given to them than to the Kaffirs, and the ration served daily is far better than that to which the coolies are accustomed in their own country. Efforts have been made to replace the rather expensive rice by mealie meal (corn meal), on which the Kaffirs in South Africa thrive, and which, as a food,

is not only cheaper but superior to rice. The Chinaman, however, has not taken kindly to mealie meal, and insists on having rice.

The following details show how the cost per shift for feeding a Chinaman is calculated:

Rice,	$0.04156	Salt,	$0.00400
Meat,	0.04302	Tea,	0.00420
Bread,	0.02324	Sundries,	0.01174
Vegetables,	0.01190	Total,	$0.13966

These data show that it is nearly twice as expensive to feed a Chinaman as it is to feed a Kaffir.

As to the relative total cost of Chinese and Kaffir labor, the Chinese for the first few months were expensive. Large capital expenditure was incurred before the arrival of the coolies, in building quarters, hospitals, etc., for their accommodation. On the French Rand mine $53,250 was spent in getting ready for the coolies. The expense of importing the laborers and returning them to China has been reckoned on an average at $87.30 per head.

When everything is taken into consideration, however, the Chinese would have worked out much cheaper than Kaffirs, had the coolies been allowed to renew their contracts for another three years, instead of being sent out of the Transvaal at the end of their first three years. From 70 to 75 per cent. of the Chinamen would have signed again had the government allowed them to exercise their right, and the saving to the mining community would have been very great.

But even in the brief period of three years Chinese labor shows up remarkably well compared with that of Kaffirs. As already noted, it takes more to feed the coolies than the Kaffirs, for the black man lives almost entirely on the cheap mealie meal (corn meal), while the coolie is given rice. The comparative total costs per shift for employing Kaffirs and Chinese are given in Table I., which also shows that so far as cost per shift is concerned, there is now not much difference between the mines run with Kaffirs and those employing Chinese. The amount of money earned per shift by the Chinaman has steadily increased. At the start he knew nothing about using a hammer in the mine, and for the first month the coolie

TABLE I.—*Comparative Total Cost of Kaffirs and Chinese per Shift for May, June, and July*, 1907.

Date.	Average No. Working.	Wages.	Food.	Hospital.	Importation and Repatriation.	Compound Expenses.	Sundry Expenses.	Total Cost.
		Robinson Gold Mining Co., Ltd. (*Kaffir Mine.*)						
May, 1907.	3,100	$0.448	$0.079	$0.010	$0.068	$0.050	.....	$0.655
June, 1907.	3,172	0.446	0.082	0.011	0.052	0.061	.....	0.652
July, 1907.	2,852	0.451	0.091	0.013	0.054	0.057	.....	0.666
		Robinson Central Deep Gold Mining Co., Ltd. (*Kaffir Mine.*)						
May, 1907.	924	0.442	0.073	0.015	0.061	0.049	.....	0.640
June, 1907.	928	0.432	0.072	0.018	0.068	0.055	.....	0.645
July, 1907.	922	0.421	0.067	0.015	0.052	0.061	.....	0.616
		New Modderfontein Gold Mining Co., Ltd. (*Chinese Mine.*)						
May, 1907.	2,059	0.358	0.125	0.013	0.104	0.074	0.002	0.676
June, 1907.	2,071	0.366	0.128	0.012	0.111	0.076	0.002	0.695
July, 1907.	2,202	0.379	0.115	0.012	0.086	0.070	0.002	0.664
		French Rand Gold Mining Co., Ltd. (*Chinese Mine.*)						
May, 1907.	1,832	0.390	0.126	0.013	0.090	0.070	0.001	0.690
June, 1907.	1,808	0.400	0.130	0.012	0.086	0.076	0.002	0.706
July, 1907.	1,784	0.430	0.132	0.012	0.094	0.069	0.007	0.744

working with hammers was paid a minimum of 25 cents per day. Then he was put on contract at 1 cent per inch drilled, and to-day the average at many mines has increased to from 45 to 55 in., instead of 36 in., as we formerly had with Kaffirs before the arrival of the coolies. Some of the best Chinamen drill as much as 100 in. per day.

As they became more efficient the Chinamen earned larger wages per shift, since most of the work is done on contract. This increased wage earned makes the total cost per shift higher than it would be were the Chinamen receiving a fixed amount per day, instead of working on contract, as the majority of them are.

But the fairest comparison is that of the cost per ton crushed on the mines run by Kaffir and by Chinese labor. Table II. shows the results obtained on the French Rand, three months before the arrival of the Chinese, and during their period of service on this mine. The third quarter of 1904 was just before the arrival of the Chinese, when Kaffirs were employed. The first batch of coolies arrived on this mine in October, 1904, and very few Kaffirs were left on the property. The data in Table II. suffice for a comparison. They show how expensive the Chinese were at first, and that there has been a steady improvement since their arrival.

On the whole, one can say that even now the coolies are just about as economical as the Kaffirs. Could they remain, the coolies would work out much cheaper.

Experience has shown that a Chinaman working underground on day's pay can give the Kaffir points in the art of loafing. Whenever possible, the coolie must be put on contract-work to obtain the best results. Under this condition he is better than the Kaffir, especially on hammer-work. So expert have some of the "hammer coolies" become, after three years' work, that to-day few of the best white miners could excel the most efficient Chinamen in using the single-hand hammer. In all drilling-contests that I have heard of along the Rand, the Chinaman has defeated his Kaffir antagonist. Little or no double-hand hammer-work is done on the Rand by the coolies.

About machinery and the stamp-mills I have found the Chinaman preferable to the Kaffir. The far higher intelligence

TABLE II.—*French Rand Gold Mining Co., Ltd., Quarterly Statement, July, 1904, to June, 1907.*

Period.	Employees. Average No. of Whites.	Natives.	Chinese.	Ratio of Colored to Whites.	Development. Footage.	Cost Per Foot.	Tons Developed.	Tons Mined.	Percentage Sorted.	Tons Milled.	Total Revenue at Mine. Value.	Per Ton Milled.	Working Expenses. Cost.	Per Ton Mined.	Per Ton Milled.	Profit. Total.	Per Ton Milled.
1904.																	
Third quarter	176	685	.....	3.886	2,033	$12.79	53,265	28,877	13.234	25,050	$201,880.76	$8.059	$163,354.79	$5.657	$6.521	$38,525.97	$1.538
Fourth quarter.....	216	152	1,214	6.307	2,311	14.52	65,349	27,905	19.369	22,500	144,775.41	6.434	203,709.21	7.300	9.053	(58,933.80*c*)	(2.619*c*)
1905.*a*																	
First quarter.....	205	154	1,382	7.468	2,021	14.58	55,520	47,728	19.439	38,450	261,630.82	6.804	252,667.05	5.294	6.571	8,963.77	0.233
Second quarter.....	216	121	1,744	8.634	2,378	11.08	47,783	52,765	21.766	41,100	310,735.62	7.560	277,785.69	5.265	6.758	32,949.93	0.802
Third quarter *b*.....	231	106	2,262	10.224	2,736	13.44	43,728	60,947	21.407	47,600	307,429.86	6.459	305,501.01	5.012	6.418	1,928.85	0.041
Fourth quarter.....	214	81	2,117	10.258	3,444	12.73	85,784	65,500	18.519	53,400	359,214.28	6.727	314,206.76	4.797	5.884	45,007.52	0.843
1906.																	
First quarter.....	215	26	1,937	9.116	3,857	11.33	69,559	64,989	19.602	51,900	335,866.86	6.471	294,081.69	4.526	5.666	41,785.17	0.805
Second quarter...	218	19	1,944	8.995	3,483	11.14	68,519	68,694	21.391	54,600	375,193.09	6.872	318,597.47	4.638	5.835	56,595.62	1.037
Third quarter.....	217	9	1,909	8.842	3,360	11.45	80,201	71,968	24.550	54,300	395,117.86	7.276	321,644.24	4.469	5.923	73,473.62	1.353
Fourth quarter.....	231	9	1,861	8.075	3,304	9.92	72,184	74,140	24.265	56,200	394,090.63	7.012	320,257.14	4.320	5.698	73,833.49	1.314
1907.																	
First quarter.....	232	15	1,832	7.931	3,471	9.13	62,425	68,160	20.848	53,850	373,673.10	6.939	307,961.90	4.518	5.719	65,711.20	1.220
Second quarter.....	204	9	1,808	8.891	3,028	10.76	51,394	65,532	17.063	54,400	359,769.12	6.613	299,104.35	4.564	5.498	60,664.77	1.115

a In February, 1905, the development redemption charge was reduced from 5s. to 3s. per ton milled.
b The small profit for this quarter is explained by the fact that new coolies arrived.
c Loss.

of the yellow man makes him superior to the black in all occupations requiring skill. In places where brute force counts for most, as in tramming and shoveling underground, I consider that the Kaffir is better than the Chinaman.

Having been "done down" scores of times during the past three years by the "wily Chinese," I have come to the conclusion that managing Chinamen is quite a different proposition from managing Kaffirs. They seem to get ahead of every plan one devises. After working a few months with Chinamen you appreciate more than ever Bret Harte's lines:

> "That for ways that are dark,
> And for tricks that are vain,
> The heathen Chinee is peculiar."

Speaking briefly of the system of the Chinese time-keeping, it is necessary to have three time-keepers for 2,000 coolies, in order to prepare the statistics required by the government, to keep up the routine-work, etc. Several systems of time-keeping are in vogue on the Rand, and the following has proved satisfactory:

Each coolie is known by a mine-number, and not by his name. On commencing work he is given a small book of 30 tickets, similar to the form shown in Fig. 5. At the end of

F.R.G.M.Co. 1			F.R.G.M.Co. 1		
Reg. No.			Reg. No.		
Class of Work			Class of Work		
M'chineH'ls No.	Drilled. Inches.	Rate.	Machine Holes. No.	Drilled. Inches.	Rate.
Boss			Boss		
Date			Date		

FIG. 5.—FORM OF TICKET USED FOR KEEPING THE TIME OF THE COOLIES.
(The dotted vertical line represents the perforations.)

the day's work the white boss fills up the details on the double ticket. One part of the ticket is torn out by the white man, the coolie keeping the duplicate in his book. When he comes from the mine the white man turns in these coolie tickets with his own at the time-office. The time-keepers then enter the amounts on these small tickets in the large time-registers under the different numbers. When the coolie's book is used up, he is handed a large pay-ticket, and is given a new book of tickets. On presenting the pay-ticket on pay-day, the Chinaman is handed his money, less any deductions, such as advances made to relatives in China, etc. Payments are made weekly, about one-quarter of the coolies being paid for their month's work every Thursday.

Most classes of labor give trouble on pay-day, as far as "loafing" is concerned, but, on the whole, the Chinese spend their money without much fuss, and lose very little time on pay-days. Their two principal vices—opium-smoking and gambling—did cause a lot of trouble at first; but the government has passed such stringent measures against these pastimes that the coolies are cautious, and do not lose very much time through these causes.

The percentage of coolies at work at the French Rand mine during the first six months of their sojourn was 89.57 per cent., while during the last six months it was 93.11 per cent. These figures compare well with those of some Kaffir mines, which show a working efficiency of about 89 per cent.

An aspect of the Chinese question seldom considered is the effect of their sojourn in the Transvaal on the coolies themselves. On the whole, their experience in South Africa has been of inestimable benefit to the coolies. The first lot of 1,400 that arrived at the French Rand mine were a motley crew. Fully 10 per cent. were criminals, and 14 per cent. were professional beggars in China. They soon found that if they were to remain in the Transvaal they must work, and men who, in China, did nothing but loaf, became in time efficient laborers. The plain, wholesome food built up many of them physically. When they return to China many of them will be skilled miners, both in hand- and machine-stoping, and, through their knowledge acquired on the Rand, will be able to command higher wages in the work of developing the mineral resources

of North China than they could ever make had they never come to the Transvaal.

True, there have been numerous outbreaks in the compounds, which, at first sight, contradict the statement that the coolies have found the mines congenial. None of these outbreaks were due to untair treatment on the part of the mining authorities or to the small pay earned. The coolies themselves say that they have not had such favorable terms before. When an industrious Chinaman can earn from $18 to $25 per month he cannot complain. The outbreaks in the compounds have been occasioned by secret societies, misunderstandings, and squabbling among themselves.

From all appearances the fate of the Chinese on the Rand is sealed. The governments in England and the Transvaal seem irrevocably determined to get rid of every coolie on the mines as soon as his first term of three years expires. Ever since the politicians in England and in South Africa commenced tinkering with the supply of labor for the mines, public confidence has fallen away, so that at present (1907) there is the severest depression ever known in South Africa.

The government confidently predict that sufficient Kaffir labor can be found to supply the mines, but all past experience is against this view, and the probability is that a year or so after all the Chinese are sent away the industry will contract unless other indentured labor is obtained. The output of gold will be in fits and starts, depending on the erratic supply of Kaffir labor, but, on the whole, there can be no great expansion. Although labor-saving devices will diminish the demand for Kaffirs somewhat, the mining conditions on the Rand are such as always to require a large army of laborers. Yet there are prophets on the Rand who declare that we have not seen the last of the Chinese. The government will send them all away, to be sure, but in three or four years' time, if other industries, such as that of the base metals, show signs of expanding, there will be such an unprecedented demand for labor that the government will be compelled to allow a limited number of coolies—say 60,000—to work the mines of the Rand.

It is not certain, however, that China would consent to coolies indenturing for the Rand again. On the whole, the Chinese consider they have been treated very shabbily in South Africa.

Not only have they been denounced by the politicians, but their contracts have been disregarded, in that they are not allowed to renew them. Many of them are aggrieved at being forced to leave the mines in defiance of their rights given in the contracts with the mining companies.

The experience with the Chinaman on the Rand goes to prove that he is one of the best rough laborers in the world, if properly handled. Great undertakings, such as the Panama Canal, could not do better than follow the Rand's example and indenture coolies for rough work.

In conclusion, I beg to express my indebtedness to Messrs. Sidney J. Jennings and C. J. Price, for permission to publish this paper.

The Treatment of the Gold-Ores of Hog Mountain, Alabama.

BY T. H. ALDRICH, JR., BIRMINGHAM, ALA.

(Chattanooga Meeting, October, 1908.)

THIS paper is intended only to give a preliminary account of experiments made, and conclusions reached, concerning the treatment of certain refractory low-grade gold-ores, the profitable reduction of which may furnish a useful guide to the solution of similar mining and metallurgical problems, not only in the gold-bearing belt of the Southern United States, but perhaps in other regions also. It must be regarded as simply a "report of progress," preceding a more elaborate and conclusive report of regular practice.

The mines of the Hillabee Gold Mining Co. are located upon what is generally known locally as Hog Mountain, on the northern edge of Tallapoosa county, Ala., about 15 miles from Goodwater and 14 miles from Alexander City, stations on the Central of Georgia railway.

Gold has been known since 1839 to exist on this property, where it was discovered in that year by a man named Johnston. The property was worked for placer-gold by the old miners, and there are now the remains of four attempts to operate it by stamp-mill and plate-amalgamation. The present owners purchased it in 1896, and since that time they have experimented at frequent intervals, in order to devise a process that would satisfactorily recover the gold. In 1903 regular operations were begun.

Geologically, the mountain consists of an intrusion of what has been pronounced true granite in altered sedimentary slates. The cooling of the granite produced shrinkage-cracks, which have become veins. There is considerable question as to the period of the slate formation; but it is undoubtedly, in my opinion, of Carboniferous age. The veins, about 50 in number, strike N. 60° to 80° E.; dip approximately 65° from the horizontal; vary from a few inches to 25 ft. (being in gen-

eral about 8 ft.) in thickness; and have an average length along the strike of about 800 ft. The vein-filling is almost pure quartz, carrying about 6 per cent. of pyrites. For about 50 ft. from the surface, down to about water-level, there is a zone of thorough decomposition, containing practically no sulphur. Below this there is a zone of about 10 ft. of partial decomposition, followed by unaltered ore which extends to an indefinite depth. Numerous geologists have established the fact that there has been great surface-erosion in this locality. Since the formation of these veins, probably at least a mile of their depth has been removed. Since this process has taken away the secondary concentrations near the original surface, the vein-material is, as might be expected, of low grade. In other words, we are mining the roots of veins, the best portion of which has long since disappeared.

The red or oxidized zone carries its gold approximately as one-third "free-milling" and two-thirds combined in some unknown way. Stamp-mill experiments have all failed. For example, 10 tons of $12 surface or red ore being treated at a well-known testing-plant, the plates caught $1.50 and the concentrating-tables $2.50, leaving $4 in the tails and $4 in the slimes that did not readily settle from the water. Under such circumstances a mechanical process was out of the question, and it was not until the advent of the cyanide process that these low-grade ores could be worked at all. Chlorination in bulk was too expensive, and unsatisfactory besides.

The average value of all the red ore upon the mountain is approximately $4 per ton, and the average value of all the hard or blue ore treated thus far is a little greater.

Experiments were begun in 1902 with the cyanide process, the result being eighty-odd per cent. extraction on $4 ore, with a consumption of cyanide of less than 0.5 lb. per ton; the time of treatment by experiment was about four days. These results being regarded as very satisfactory, operations were begun upon the property in the spring of 1903. The principal difficulty experienced was in making the results of practice conform to those of experiment.

Fine grinding and continuous leaching did not give a good extraction; and after several thousand trials the process finally adopted was dry-crushing to pass a screen with $\frac{1}{8}$- by $\frac{1}{2}$-in. slots;

and then leaching intermittently by repeatedly flooding the top of the ore to 1 ft. depth of solution as quickly as possible, allowing this to leach through rapidly, and draining dry for the remainder of 24 hours. Experiments have shown that the solution of the gold takes place principally when the ore is nearly dry. With fine grinding and continuous leaching the extraction was 50 per cent., whereas, with coarse grinding and intermittent leaching it was 90 per cent. in considerably reduced time, with a smaller quantity of solution, and much cheaper grinding. This action seems to be entirely mechanical and not to have any relation to oxidation or aëration of the ore. The following illustration may explain it: If, of two sponges which have been dipped in a stained solution, one be suspended in a gently moving stream with flow-velocity of, say, 5 in. per hour, and the other put into clear water, raised out of it, and drained dry several times, the latter would be washed cleaner in less time and with considerably less water. In the case before us, the intermittent leach permits the water to go in and out of every pore and crevice, carrying fresh solution in and out of the pores, and removing the products of the chemical reaction which tend to plug them.

The amount of ore upon the property is approximately 500,000 tons to every 100 ft. of depth. With but 250,000 tons of surface-ore (a large amount of this loss is in mining because of the rotten walls), it became necessary, as soon as the red-ore process was once well established and running, to attack the problem of the hard blue ores which are the main future supply. The treatment of the surface-ores, although it took considerable time to work out, was very easy compared with that of the blue ores. The former were mined entirely by hand, whereas, with the blue ores, the first difficulty encountered was the almost impossibility of drilling the material. We have tried several makes of hammer-drills and piston-drills, using several sizes of each. The rock is so hard and tough that the steel will not disrupt the particles unless driven with a great deal more force than is usually the practice. For example, a hand hammer-drill similar to the Hartsoeg "Little Wonder," which will drill most rocks, in this case simply bounces back, and actually as much (if not more) is cut from the bit as from the rock. Again, when the rock is blasted,

it comes down in very large pieces, which are so difficult to break up afterwards that it would cost more to "bulldoze" these large lumps than the ore is worth. These two obstacles seemed insurmountable at the outset, but we have satisfactorily overcome them.

Numerous experiments with drill-steels showed that, on our ore, one steel was about as good as another. The only one showing any marked improvement over the others was vanadium-steel, which gave better results simply because it was stronger. We use a $2\frac{3}{4}$-in. piston-type with as small bit-gauge as possible, and drill at 110 lb. air-pressure; and by thus concentrating a heavy blow upon a small cutting-edge we are able to drill down-holes satisfactorily with a piston-drill. The up-holes and stope-holes we drill with the 3-in. Leyner type, making the bit as small as possible, and thus concentrating the blow. We use 12-in. changes on the Leyner and 20-in. on the piston-type. It is very difficult to get the bits to drill their full change. The wear and tear on the drills is in favor of the Leyner, as is also air-economy, while the cost of steel and blacksmithing is in favor of the piston-drill. In down-holes the piston-drill seems to bore a little faster than the other. The up-holes, on the contrary, the piston-drill will not make at all. No matter how slowly the drill is run or with what care, the cutting-edge heats, loses its hardness, and drives back.

We use 1 in. and $1\frac{1}{8}$ in. of 75-per cent. gelatine dynamite, made especially for us by the Ætna Powder Co., which has a peculiar quality of giving the necessary "fragmentation" to the rock. The 75-per cent. nitroglycerine dynamite will shoot the rock as well as the Ætna gelatine; but the bulk of the ore broken from an 8-ft. vein (taking on 6 ft. of ground with a 7-ft. hole) will be in pieces larger than 2-ft. cubes; whereas this gelatine dynamite (even less of it being used than of the glycerine under the same conditions) throws nothing as large as a 1-ft. cube. In other words, the whole product is of crusher-size and there is no hammer-work whatever.

The next problem was the extraction of the gold. The blue ore, being very hard and brittle, was comparatively easy to mill, although with a larger consumption of power and wearing-parts. The capacity of the mill remained substantially the same as with red ore—if anything, a little greater. Leaching the

blue ore exactly as we had leached the red ore gave but 50 per cent. extraction. Fine grinding, so that the whole product would pass 200-mesh, and agitating with air, extracted generally $3 from $4 ore; sometimes agitation for 2 hr. would leave only 40 cents per ton; at other times we could not get the tailings below $1.75, even by agitating for 10 hr. This process, although it worked fairly well, was not commercially successful, because of the high tail-values on the low-grade ore, and the contamination of the cyanide solution due to the reducing agents, which soon rendered it inert.

This process was tried for about six months on about 1,000 tons a month, and finally abandoned in favor of a system of dehydration-roasting which consists in heating the ore to about 300° F. A "dead sweet" roast, either with or without salt, was very much more expensive, and did not give as good results. Microscopic examination shows the ore to contain a great many small inclusions of liquid, gas, and vapor. Upon heating, these inclusions expand, and disrupt their walls, thus rendering the ore porous, as it were, by a series of microscopic explosions. This gave on $4 material passing $\frac{1}{8}$- by $\frac{1}{2}$-in. slots an extraction of 75 per cent. It was cheaper and better than fine grinding, because the reducing agents were largely destroyed, and the roasting was very cheap. We used a rotary kiln with tube 4 by 40 ft.; and two men roasted and charged into the leaching-tank 50 tons per shift of 12 hr., with a consumption of 1.5 cords of wood. Yet even this was not commercially successful. Something better had to be devised to make these low-grade ores profitable.

Fine grinding and agitation in hot cyanide solution carrying 5 lb. of magnesium oxide per ton, gave an extraction of 90 per cent. on $4 ore, or 60 cents per ton better extraction than either of the foregoing methods; but the cyanide-consumption was a little higher, and the precipitation on zinc-shavings of magnesia-lime solution was impracticable, one crowding the other out of solution and fouling the zinc. We could not use magnesia alone because of the treatment of the red ores in the same plant.

I then began to study chemically and with a microscope the exact manner in which the gold was carried in the ore and how it was locked up. After a year and a half of diligent

research it was finally established: (1) that all the gold is metallic, and in isolated metallic particles; (2) that the "locking-up" substance is probably a compound of silver, practically insoluble in cyanide. The ore will not show "colors" in a pan, or yield to amalgamation or concentration in water, and therefore gold is disseminated through it independent of the pyrites. All our experiments seemed to bring the extraction to a certain point and then stop. We could leach average ore to $1 a ton, $4 ore to $1, and $1.50 ore to about 90 cents, no matter by which process treated. That last dollar seemed to be hopelessly locked up.

We have now completed a plant to operate on an entirely new principle, by amalgamation with copper sulphate, salt, iron, and mercury, similar to the Kröncke and *patio* processes, in a large 5- by 22-ft. tube-mill, the grinding, chemical treatment, and amalgamation going on simultaneously. The mercury is badly fouled and floured in this operation; and upon being discharged from the tube-mill, the pulp receives an additional treatment by the addition of a cleansing solution, and passage of the whole over plates. In experiment, this process has given very satisfactory results, bringing $3.60 ore down to 20-cent tail-values, and is very much cheaper than cyaniding could possibly be. We estimate that, besides recovering 80 cents more per ton from the ore, it will be 50 cents per ton cheaper to operate. If the figures of practice substantiate those of experiment, it will be the final solution of our problem. However, it is too soon to say much about the process, as no data from practice are at hand.

The plant is, however, operating steadily with the red ore, and intermittently by the dehydration process with the blue ore, when the value of the latter is sufficiently high to make it profitable. The low-grade blue ore is left in the mines.

The work was done with the assistance of my partner, A. P. Kennedy, other metallurgists, and the U. S. Geological Survey; and it is our intention later to publish the complete data which have led to the development of a process by which this very hard and very refractory ore can be mined and milled at a profit when it carries less than $3 per ton.

The Constitution of Copper-Iron and Copper-Lead-Iron Mattes.*

BY CHARLES H. FULTON AND IVAN E. GOODNER, RAPID CITY, SO. DAK.

(Chattanooga Meeting, October, 1908.)

THE subject of the constitution of copper-iron mattes has received considerable attention in recent years by Keller,[1] Bolles,[2] Hofman,[3] and Gibb and Philp.[4] Still more recently Friedrich, Röntgen, Wiedmann, Heyn and Bauer, and Leroux have published, in some instances, elaborate researches on the various systems of the sulphides, and the sulphides and their respective metals, which enter into the constitution of mattes, and have added very greatly to the knowledge of the subject. The work of Friedrich has been especially fruitful. But to date no effort has been made to correlate the data obtained in the latter researches and apply them to the study of the product called "matte," which is made in such large quantities by copper- and lead-smelters.

In this paper we propose to determine, by metallographic and chemical means, the constitution of copper-iron and copper-lead-iron mattes in the light of the recent data above mentioned. We do not discuss copper-nickel mattes or speisses (artificial arsenides).

The paper is divided into three sections:

I. A review and discussion of the various binary systems of the metallic sulphides, and the binary systems of various metals and their respective sulphides entering into the constitution of mattes; also a discussion of the physical properties of these sulphides.

II. A description of the results obtained from the chemical and microscopic examinations of the samples of copper-iron and copper-lead-iron matte obtained from various plants.

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* A contribution from the Metallurgical Laboratory of the South Dakota School of Mines, Rapid City, So. Dak.

[1] *Mineral Industry*, vol. ix., p. 240 (1900).

[2] *Trans.*, xxxv., 666 (1905).

[3] *Trans.*, xxxviii., 142 (1908).

[4] *Trans.*, xxxvi., 665 (1906).

III. A discussion of the constitution of mattes, based on the data of the first two parts of this paper.

I. The Various Binary Systems of the Metallic Sulphides, and the Binary Systems of Various Sulphides and Their Respective Metals Entering into the Constitution of Mattes.

The chief sulphides found in mattes are cuprous sulphide, Cu_2S; ferrous sulphide, FeS; and lead sulphide, PbS. Zinc sulphide, ZnS, is present in subordinate amounts; silver is present, probably as sulphide Ag_2S, and usually in very small amount only. The condition of the gold is not definitely known.

The metals present in the pure state, or nearly so, are iron and copper. Other impurities, such as antimony and arsenic, are present in most mattes, but in very small amounts, and their condition is not considered in the present paper. The sulphides enumerated above are considered as chemical compounds, though, judging from their thermal analysis, Friedrich[5] does not believe this theorem to be absolutely established.

The binary systems of the sulphides discussed are: FeS-Cu_2S; PbS-Cu_2S; Ag_2S-Cu_2S; ZnS-Cu_2S; FeS-PbS; Fe_7S_8-PbS; FeS-ZnS; PbS-Ag_2S; PbS-ZnS; and Ag_2S-ZnS; and the systems of metals and their respective sulphides are: Fe-FeS; Cu-Cu_2S; and Pb-PbS.

What Are the End Members of the Sulphide Series, and What Are Their Melting-Points?

It has been mentioned in this paper that the sulphides are considered as chemical compounds, but the end members of the various binary systems are rarely the pure compound; in fact, it will be shown that the end member may consist of the sulphide plus either sulphur or the metal in a small excess. Thus, the iron sulphide member may be FeS + Fe or FeS + S; the cuprous sulphide member, Cu_2S + Cu or Cu_2S + S, etc.

1. *Iron Sulphide.*—The subject of iron sulphide has been thoroughly investigated by Le Chatelier and Ziegler,[6] who came

[5] *Metallurgie*, vol. v., No. 2, p. 57 (Jan. 22, 1908).

[6] *Bulletin de la Société d'Encouragement pour l'Industrie nationale*, vol. ciii., p. 368 (September, 1902).

to the conclusion that the proto-sulphide of iron, FeS, is a chemical compound, and that no artificial sub-sulphide of iron exists. This has also been demonstrated by other investigators. Le Chatelier and Ziegler show that at a temperature of 1,200° C. the compound FeS begins to lose sulphur by vaporization, the result being a mixture of FeS and Fe, and that at 1,500° C. this decomposition is very appreciable. They also show that iron oxide, probably Fe_3O_4, may be present in the sulphide and will form a eutectic with iron sulphide. Thus, ordinary iron sulphide is a ternary system, showing under the microscope FeS and Fe and a eutectic of FeS and Fe_3O_4. (See remarks on

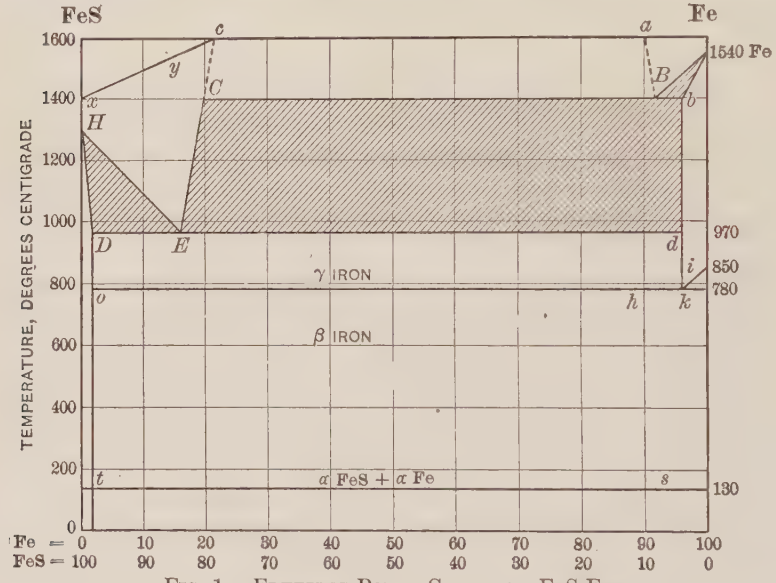


FIG. 1.—FREEZING-POINT CURVE OF FeS-Fe.

ternary systems further on in this paper.) Treitschke and Tammann[7] have also investigated the system FeS-Fe, and confirm the results of Le Chatelier and Ziegler. They give the melting-point of FeS by extrapolation at 1,300° C., which is probably high, as is shown later in this paper.

The eutectic between Fe and FeS has a composition of 16 per cent. Fe and 84 per cent. FeS, and freezes at 970° C. The components of the eutectic, however, are not pure Fe and FeS, but mixed crystals of FeS dissolving 2 per cent. Fe, and Fe dissolving a small proportion of FeS. (Hereafter in the paper we

[7] Über das Zustandsdiagramm von Eisen und Schwefel, *Zeitschrift für anorganische Chemie*, vol. xlix., No. 3, p. 320 (May 31, 1906).

will use the term "mixed crystal" in preference to "isomorphous mixture" or "solid solution." As the term "solid solution" strictly implies a substance not crystalline, but amorphous, we prefer the term "mixed crystal" to it.) The curve reproduced in Fig. 1 shows that the two components are largely insoluble in each other even at the melting-point of iron. This type of curve is discussed by Howe under type case 5 Bβ.[8]

In the mattes produced in copper- and lead-smelting the FeS of the matte is, in most cases, derived from some higher sulphide, FeS_2, pyrite; Fe_7S_8, pyrrhotite; or $CuFeS_2$, chalcopyrite, or other copper-iron sulphides. As the temperature of smelting almost invariably exceeds 1,200° C., the final product, as regards the iron sulphide, is a member of the series FeS-Fe, the amount of metallic iron present depending upon the temperature reached. The vaporization of sulphur from FeS increases rapidly with the temperature, but is not great until 1,400° or 1,500° C. is reached, so that the amount of metallic iron in most mattes is not very great, and is usually in solution in the FeS, and possibly in the Cu_2S. Some mattes, however, produced at a high temperature may have a large quantity of metallic iron present. That iron oxide, Fe_3O_4, may be present with iron sulphide is shown by the researches of Le Chatelier and Ziegler, and confirmed by Treitschke and Tammann. Iron oxide and iron sulphide can be coexistent under a temperature of about 1,100° C.,[9] but with certain concentration (high FeS) and temperatures above 1,100° or 1,200° C., the two substances react, with the production of metallic iron and SO_2:

$$5\ Fe + 2\ SO_2 \leftrightharpoons 2\ FeS + Fe_3O_4.$$

We have failed to find any definite microscopic evidence of iron oxide in mattes, although it is evident that, under certain conditions, its existence is theoretically possible. The most favorable conditions for the production of mattes containing iron oxide are to be found in reverberatory smelting.

The system FeS-S has not been extensively investigated, but from the references in Table I. it is shown by independent observers that the melting-points of some members of the series lie above that of FeS. The melting-point of pure FeS can be

[8] *Iron, Steel, and Other Alloys*, p. 132 (1903).

[9] Heyn and Bauer, *Metallurgie*, vol iii., No. 3, p. 84 (Feb. 8, 1906).

obtained practically only by extrapolation or interpolation, in so far as heating above its melting-point, which is necessary to make the determination, causes some decomposition. Hofman[10] places the melting-point of FeS at 975° C., but judging from his metallographic work his material is evidently not pure iron sulphide.

TABLE I.—*Melting-Point and Composition of the End Members of the FeS-S and FeS-Fe Systems.*

Substance.	Melting-Point.	Chemical Composition.		Reference.
		Fe, Per Cent.	S, Per Cent.	
FeS.....	1,171°C.	63.59	36.41	Friedrich, *Metallurgie*, v. 5, p. 56.
FeS.....	1,194°C.	63.59	36.41	Borneman, *Metallurgie*, v. 5, p. 64.
FeS + Fe.	1,189°C.	63.70	36.21	Borneman, *Metallurgie*, v. 5, p. 64.
FeS + S...	1,203°C.	63.25	36.75	Borneman, *Metallurgie*, v. 5, p. 64.
FeS + S...	1,192°C.	61.75	38.25	Friedrich, *Metallurgie*, v. 5. p. 56.
FeS?.....	1,133°C.	?		Röntgen, *Metallurgie*, v. 3, p. 479.
FeS?.....	1,300°C.	?		Tammann, *Zeitschrift für anorganische Chemie*, v. 49, No. 3, p. 320.

According to Le Chatelier and Ziegler, and Treitschke and Tammann, iron sulphide has a transformation-point at 140° C., with possibly a second point somewhat higher. This dimorphic change is accompanied by a contraction and shrinkage on cooling, and accounts for the fissured and porous appearance of this sulphide in mattes.

From the foregoing it would seem that the end member as regards iron sulphide is apt to be either FeS containing some Fe, or FeS containing some S, the iron or sulphur in excess being in solution in the FeS if present in not too great an amount. Considering the temperature of the production of mattes, which is almost invariably above 1,200° C., and the sulphur-content of matte as determined by chemical analysis, the iron sulphide of the matte is without question a member of the series FeS-Fe. We examined some mattes which contained much metallic iron. These were produced by partial pyrite-smelting, with a slag of from 48 to 49 per cent. of SiO_2, requiring a high temperature for fusion.

These facts also explain the production of the so-called "iron sows" found most frequently in partial pyrite-smelting with highly siliceous slags requiring high temperatures. The sows represent the segregated iron from the matte.[11] It is also prob-

[10] *Trans.*, xxxviii., 137 (1908).

[11] See also Peters's *Principles of Copper Smelting*, p. 403 (1907).

able that at these high temperatures the reaction between FeS and Fe_3O_4 in the presence of much FeS contributes to the formation of metallic iron.[12]

2. *Cuprous Sulphide.*—This substance presents the same general aspect as iron sulphide. It is a chemical compound of the formula Cu_2S, and is the only sub-sulphide known as far as the researches go.[13] Fig. 2 shows the curve of the system Cu_2S-Cu.

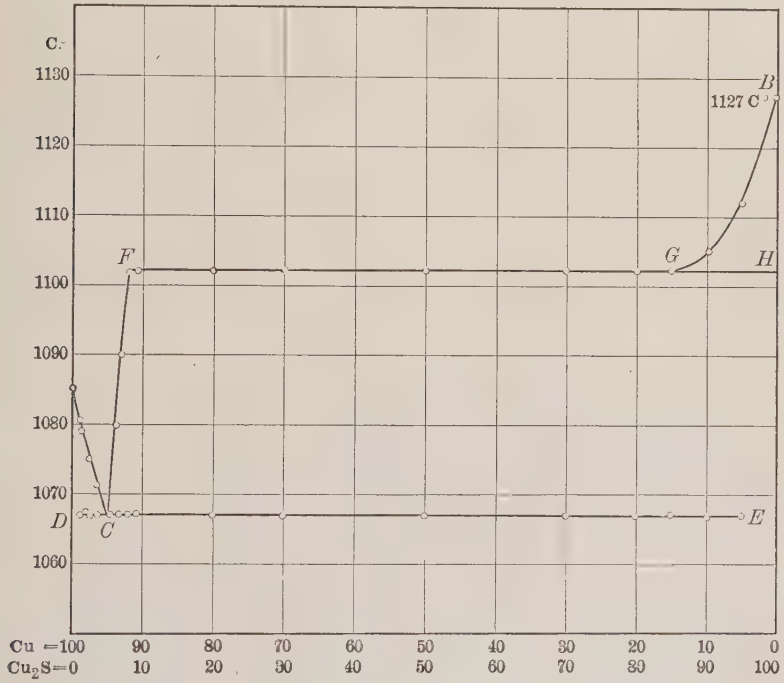


FIG. 2.—FREEZING-POINT CURVE OF Cu_2S-Cu.

It is of the same type as that of FeS-Fe, except that in this case the eutectic lies near the metal side, and has a composition of 3.8 per cent. of Cu_2S and 96.2 per cent. of Cu, and freezes at 1,067° C. Cu and Cu_2S are practically insoluble in each other in the solid state. An addition of either sulphur or copper to the sulphide has the effect of lowering the melting-point. Table II. gives data on the melting-point of cuprous sulphide.

[12] *Engineering and Mining Journal*, vol. lxxviii., No. 9, p. 333 (Sept. 1, 1904).
[13] Heyn and Bauer, *Ibid.*

TABLE II.—*Melting-Point and Composition of the End Members of the Cu_2S-Cu and Cu_2S-S Systems.*

Substance.	Melting-Point.	Chemical Composition.	Reference.
Cu_2S + S..	1,127° C.	Cu_2S, 99.71 per cent.	Heyn and Bauer, *Metallurgie*, v. 3, p 78
Cu_2S.....	1,135° C.	Cu_2S = 100 per cent.	Friedrich, *Metallurgie*, v. 5, p. 52
Cu_2S + S..	1,101° C.	Cu = 78.6 per cent.	Friedrich, *Metallurgie*, v. 5, p. 52
Cu_2S ?	1,085° C.	?	Röntgen, *Metallurgie*, v. 3, p. 479
Cu_2S + S..	1,152° C.	Cu = 78.94 per cent.	Hofman, *Trans.*, xxxviii., 142.
Cu_2S + Cu	1,110° C.	Cu_2S = 95 per cent., Cu = 5 per cent.	Heyn and Bauer, *Ibid.*

Copper sulphide, like iron and other sulphides, has the property of losing sulphur by volatilization above its melting-point.[14] From the foregoing data, and taking into consideration the temperature of smelting in the production of mattes, and the evidence furnished by chemical analysis and microscopic researches outlined later in this paper, it is evident that the end member on the cuprous sulphide side is one of the series Cu_2S-Cu.

Cuprous sulphide undergoes a dimorphic change at 103° C.[15] At this temperature copper which has previously been in solution is probably forced out as the so-called "moss copper" of mattes. This is referred to more in detail in the last division of this paper.

3. *Lead Sulphide.*—PbS is the lowest sulphide, no sub-sulphides being known. The system Pb-PbS, which has been worked out by K. Friedrich and A. Leroux,[16] shows a curve dropping from the melting-point of the sulphide to that of lead, with no eutectic point and a practical insolubility of the constituents in each other in the solid state. Table III. gives the data of the melting-point of PbS.

TABLE III.—*Melting-Point of PbS, Etc.*

Substance.	Melting-Point.	Chemical Composition.		Reference.
		Pb, Per Cent.	S, Per Cent.	
PbS.....	1,120° C.	86.6	13.4	Friedrich, *Metallurgie*, v. 5, p. 51
PbS + S..	1,130° C.	85.9	14.1	Friedrich, *Metallurgie*, v. 5, p. 51
PbS + Pb	1,114° C.	87.1	12.9	Friedrich, *Metallurgie*, v. 5, p. 51
PbS ?.....	970° C.	?		Weidmann, *Metallurgie*, v. 3, p. 660

[14] Friedrich, Beiträge zur Kenntniss der Schwefelmetalle, *Metallurgie*, vol. v., No. 2, p. 50 (Jan. 22, 1908); also vol. v., No. 4, p. 122 (Feb. 22, 1908).

[15] Landolt and Börnstein, *Physikalisch-chemische Tabellen*, 3d ed., p. 277 (1905).

[16] Blei und Schwefel, *Metallurgie*, vol. ii., No. 22, p. 536 (Nov. 22, 1905).

PbS, like other sulphides, suffers dissociation above its melting-point and loses sulphur, and it is markedly volatile as a whole.[17] The loss of sulphur, however, is not so marked as in the case of iron sulphide and cuprous sulphide. The end member in the case of lead sulphide in mattes is probably very nearly PbS.

4. *Zinc Sulphide.*—This sulphide enters into the constitution of matte to but a limited extent. Its melting-point has not been determined, due to its marked volatility at high temperatures and to the fact that the boiling- and melting-points lie very close together. Friedrich places the melting-point by extrapolation close to 1,670° C.

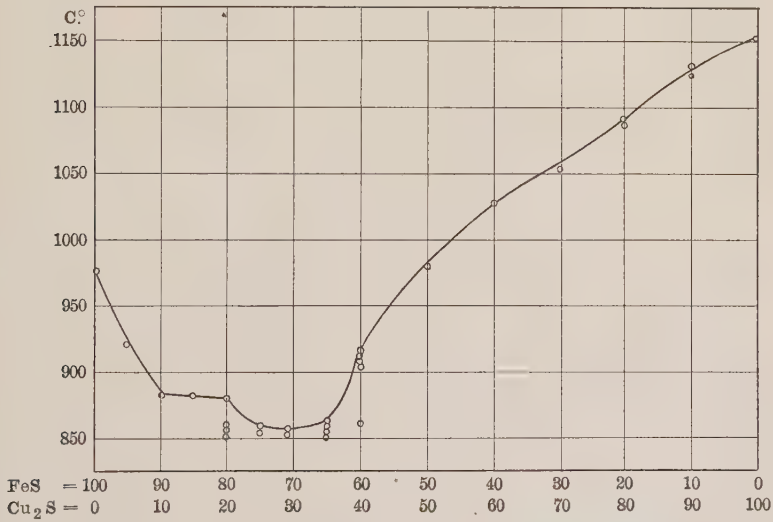


FIG. 3.—FREEZING-POINT CURVE OF FeS-Cu_2S (HOFMAN).

The Binary Systems of the Sulphides in Detail.

1. *Ferrous Sulphide-Cuprous Sulphide.*—This system has been investigated by Hofman[18] and Röntgen.[19] Fig. 3 shows the Hofman curve and Fig. 4 that of Röntgen. The first is replotted from the author's data. Both curves are rather unsatisfactory. Hofman did not use pure iron sulphide as an end member, as is shown by his microphotographs. The Röntgen curve is very com-

[17] K. Friedrich, *Metallurgie*, vol. v., No. 2, p. 51 (Jan. 22, 1908); Friedrich and Leroux, *Metallurgie*, vol. ii., No. 22, p. 536 (Nov. 22, 1905).

[18] *Trans.*, xxxviii., 142 (1908).

[19] *Metallurgie*, vol iii., No. 14, p. 479 (July 22, 1906).

plicated, and indicates with the end members the existence of five and possibly six maxima in the system corresponding to chemical compounds, but the microphotographs fail to bear out his results on the curve, in showing by structure the presence of chemical compounds except end members. Röntgen shows a main eutectic of the following composition: FeS, 67, and Cu_2S, 33 per cent., freezing at 895° C. Hofman shows a eutectic, determined chiefly by microscopic work, of FeS, 86, and Cu_2S, 14 per cent., freezing at 860° C. From our examination of copper-iron mattes we believe the eutectics of Röntgen and Hofman to be identical, and the only one occurring in the sys-

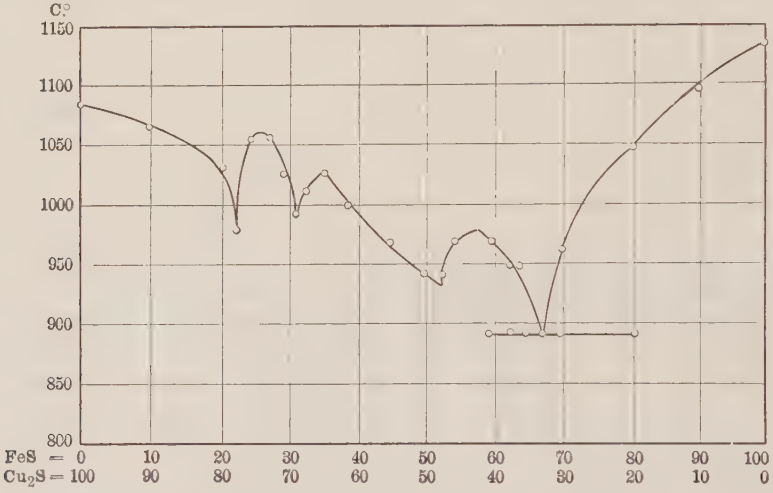


FIG. 4.—FREEZING-POINT CURVE OF FeS-Cu_2S (RÖNTGEN).

tem, and that its true composition is very close to Cu_2S, 21 to 22, and FeS, 78 to 79 per cent., which corresponds to Cu, 17 per cent.

Hofman gives the eutectiferous range as extending from 20 to 90 per cent. of FeS, assuming the existence of mixed crystals, one of 10 per cent. of Cu_2S in 90 per cent. of FeS, and the other of 20 per cent. of FeS in 80 per cent. of Cu_2S. From our own work we strongly support this view, although we place somewhat different limits, as will be seen later in this paper. In the metallographic work on mattes we have been unable to detect the chemical compounds mentioned by Röntgen. For the correct melting-points of the end members of the series, reference is made to the discussion of the respective sulphides.

2. *Lead Sulphide-Cuprous Sulphide.*—This system, which has been investigated by K. Friedrich,[20] has the freezing-point curve shown in Fig. 5. The end members in the investigation were not absolutely pure, the cuprous sulphide member being represented by Cu_2S, 99.6 per cent., and the lead sulphide member by PbS, 99.6 per cent. The series is eutectiferous throughout, implying but little relative reciprocal solubility of the constituents in the solid state. The eutectic has a composition of Cu_2S, 51, and PbS, 49 per cent., with the freezing-point at 540° C. This is the important system in lead-copper mattes.

3. *Lead Sulphide-Ferrous Sulphide.*—This system has been investigated by Weidmann.[21] The related system of lead sulphide-pyrrhotite has been investigated by Friedrich,[22] with practically

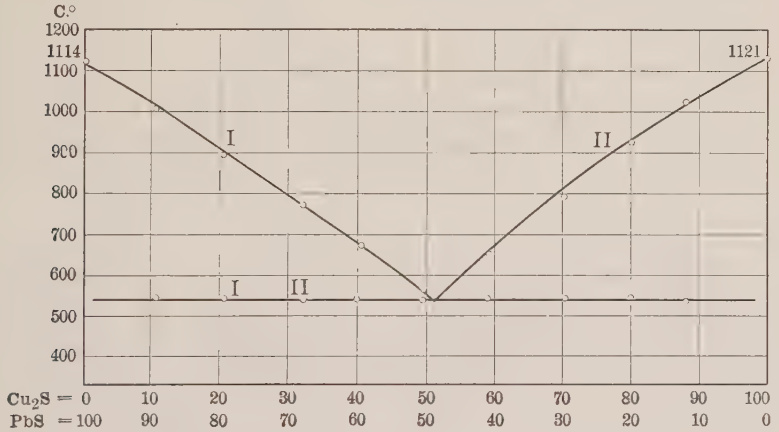


FIG. 5.—FREEZING-POINT CURVE OF Cu_2S-PbS.

the same result. The freezing-point curves are shown in Figs. 6 and 7, respectively. In neither case are the end members the pure sulphides, but they are very nearly pure.

Weidmann places the melting-point of PbS at 970° C., or 133° C. lower than Friedrich. The latter's figures, however, have been repeatedly determined and are accepted by us. While the two systems are not identical, one having for an end member FeS, and the other Fe_7S_8, approximately they may be considered to belong to the same ternary system, Pb-Fe-S, and occupy planes in this system not very far apart. They are therefore

[20] Bleiglanz-Kupfersulfür, *Metallurgie*, vol. iv., No. 20, p. 671 (Oct. 22, 1907).
[21] Bleistein, *Metallurgie*, vol. iii., No. 19, p. 660 (Oct. 8, 1906).
[22] Bleiglanz-Magnetkies, *Metallurgie*, vol. iv., No. 14, p. 480 (July 22, 1907).

approximately comparable. Weidmann places the eutectic at FeS, 25.8, and PbS, 74.2 per cent., with the freezing-point at

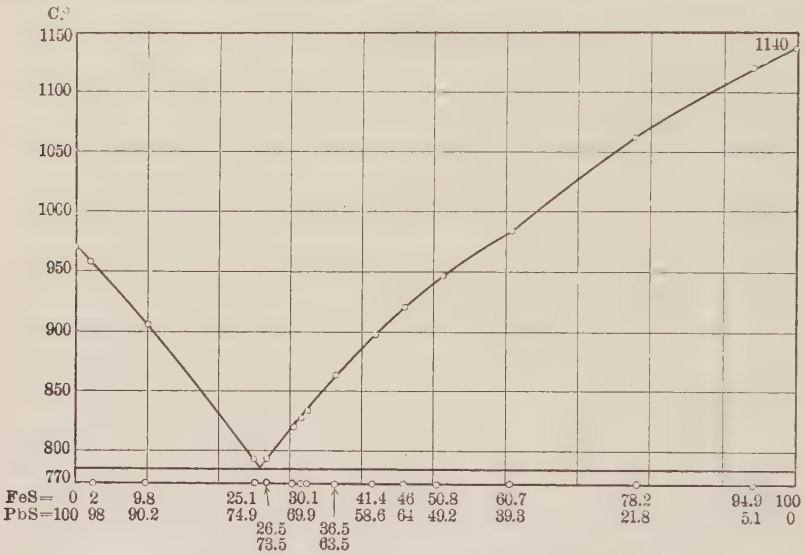


FIG. 6.—FREEZING-POINT CURVE OF FeS-PbS.

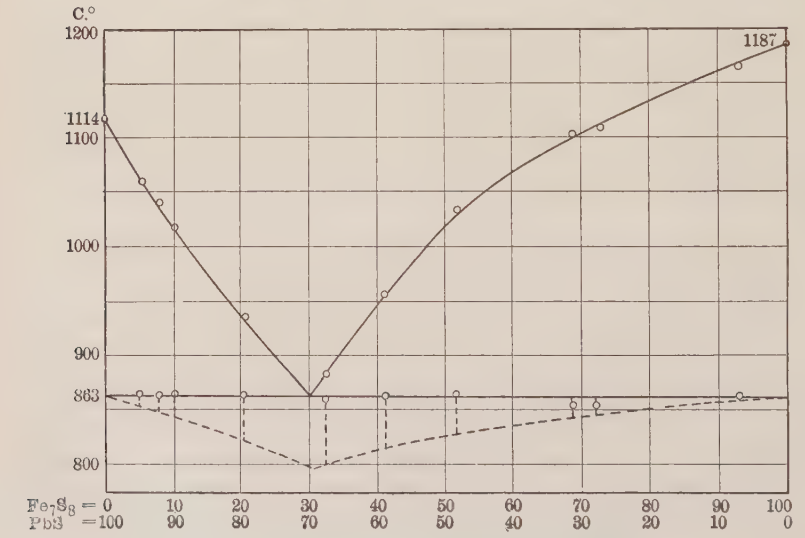


FIG. 7.—FREEZING-POINT CURVE OF Fe_7S_8-PbS.

782° C. Friedrich gives the eutectic at Fe_7S_8, 30, and PbS, 70 per cent., with a freezing-point at 863° C. Both investigators show the eutectic range to be from ordinate to ordinate, imply-

ing a practical reciprocal insolubility of the constituents in the solid state.

4. *Silver Sulphide-Cuprous Sulphide.*—K. Friedrich has investigated this system,[23] the freezing-point curve of which is reproduced in Fig. 8. The system consists of an unbroken series of mixed crystals. Neither the thermal investigations nor the microscopic sections give any indications of a eutectic. The end members were not absolutely pure sulphides. The Ag_2S had a melting-point of 835° C. and the Cu_2S of 1,121° C. The curve shows a minimum at 677° C. at a composition of Ag_2S, 70, and Cu_2S, 30 per cent.

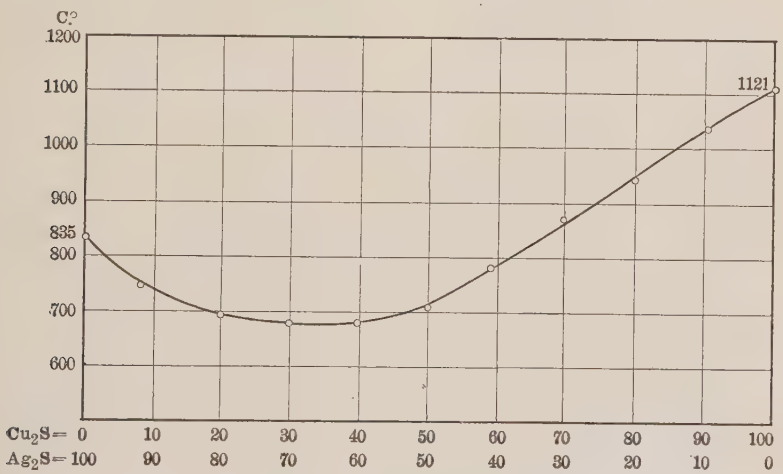


FIG. 8.—FREEZING-POINT CURVE OF Ag_2S-Cu_2S.

This is the only system of all those under discussion in which the constituents form an unbroken series of mixed crystals, implying a very powerful solvent action. There seems little doubt that in the presence of cuprous sulphide, any silver present in the matte is held in solution by this cuprous sulphide. A study of the results obtained indicates that the silver in mattes is probably present as silver sulphide, unless the temperature at which the matte is produced is unusually high.

5. *Silver Sulphide-Lead Sulphide.*—This system has been investigated by K. Friedrich[24] and is chiefly of interest in connection with the manner in which the precious metals are

[23] Schwefelsilber-Kupfersulfür, *Metallurgie*, vol. iv., No. 20, p. 671 (Oct. 22, 1907).

[24] Bleiglanz–Schwefelsilber, *Metallurgie*, vol. iv., No. 14, p. 479 (July 22, 1907).

carried in mattes. The freezing-point curve of the system is shown in Fig. 9. Neither of the end members was a pure sulphide. The melting-point of the Ag_2S was 835°C. and that of the PbS 1,114°C. A eutectic occurs of the composition Ag_2S, 77, and PbS, 23 per cent., which has a freezing-point of 630°C. The eutectiferous range does not extend across to the ordinates, but there is a reciprocal solubility of 2 per cent., forming two series of mixed crystals.

Lead sulphide has, therefore, an appreciable solvent action on silver sulphide sufficient for us to assume all of the silver in mattes to be in solution in the lead sulphide, provided cuprous

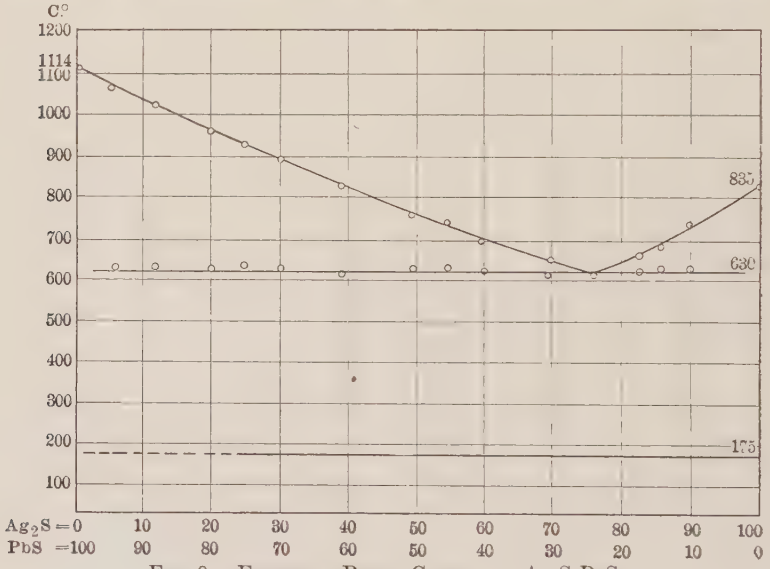


FIG. 9.—FREEZING-POINT CURVE OF Ag_2S-PbS.

sulphide is absent. The solvent action of cuprous sulphide on silver sulphide is undoubtedly more powerful than that of lead sulphide.

6. *Silver Sulphide-Ferrous Sulphide.*—This system has not been investigated, but from our experience in matte-smelting we know that ferrous sulphide has practically no solvent action on silver or on gold. Metallic iron, however, has a powerful solvent action on gold, the system iron-gold forming two series of mixed crystals.[25] Metallic iron, however, has practically no

[25] Bolles, *Trans.*, xxxv., 666 (1905). Pierce, *Trans.*, xviii., 454, 457 (1889–90). Spilsbury, *Trans.*, xv., 767 (1886–7). E. Isaac and G. Tammann, Über Eisen-Gold Legierungen, *Metallurgie*, vol. iv., No. 13, p. 469, (July 8, 1907).

solvent action on silver.[26] Cuprous sulphide and copper, as is well known, have a powerful solvent action on gold.

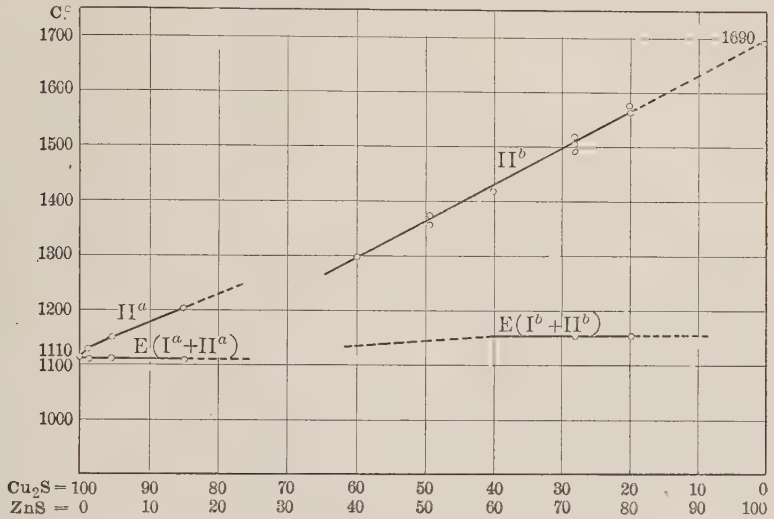


FIG. 10.—FREEZING-POINT CURVE OF Cu_2S-ZnS.

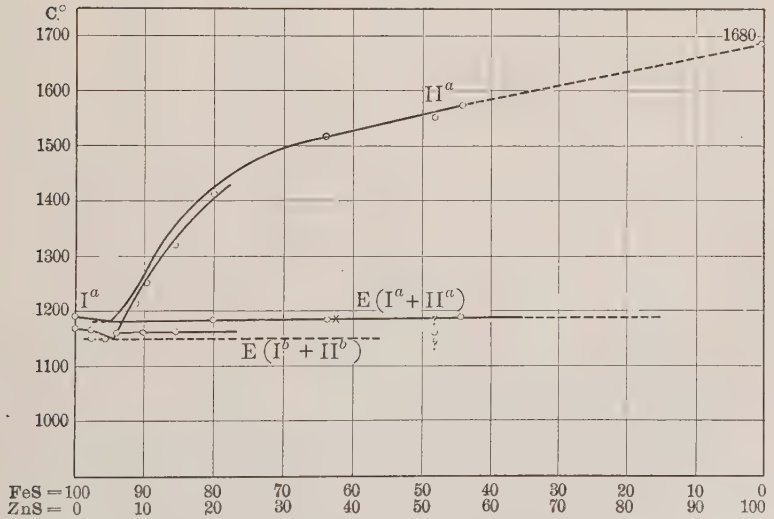


FIG. 11.—FREEZING-POINT CURVE OF FeS-ZnS.

7. *Zinc Sulphide-Cuprous Sulphide; Zinc Sulphide-Ferrous Sulphide; Zinc Sulphide-Lead Sulphide; and Zinc Sulphide-Silver Sulphide.*—These systems have been investigated by K.

[26] Bolles, *Ibid.* Pierce, *Ibid.* Spilsbury, *Ibid.*

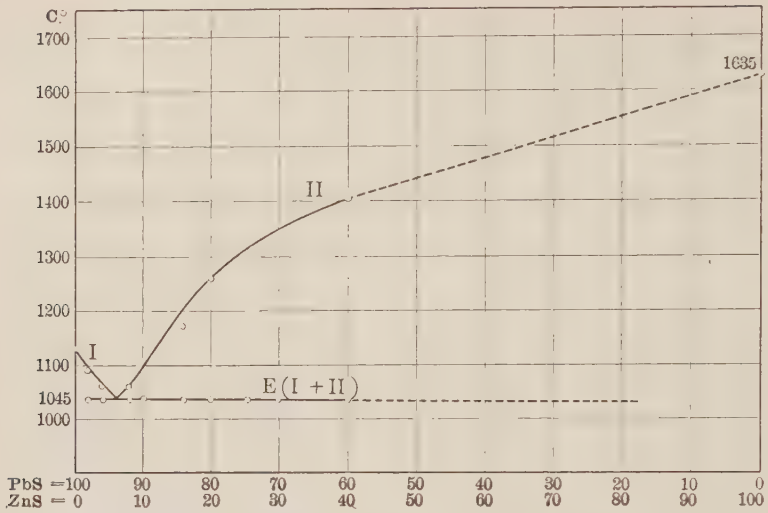


FIG. 12.—FREEZING-POINT CURVE OF PbS-ZnS.

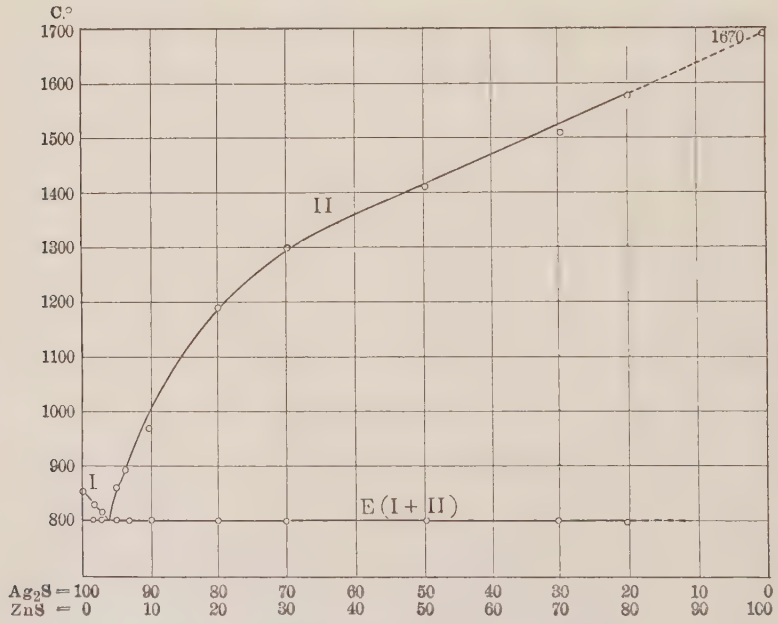


FIG. 13.—FREEZING-POINT CURVE OF Ag_2S-ZnS.

Friedrich[27] and their freezing-point curves are shown in Figs. 10, 11, 12 and 13 respectively. Zinc is frequently present in

[27] Die Zinc blende als Stein bilder, *Metallurgie*, vol. v., No. 4, p. 114 (Feb. 22, 1908).

matte as a minor constituent, as a rule rarely above 3 or 4 per cent., and most frequently less than this amount.

Zinc sulphide-cuprous sulphide form no eutectic. Zinc sulphide is practically totally insoluble in cuprous sulphide, and *vice versâ*.

Zinc sulphide-iron sulphide form a eutectic of the composition, FeS, 95, and ZnS, 5 per cent., with a freezing-point at approximately 1,145°C. Zinc sulphide possibly forms a mixed crystal with iron sulphide up to 3 per cent. Whether iron sulphide is soluble in zinc sulphide is doubtful.

Zinc sulphide-lead sulphide form a eutectic of the composition PbS, 92, and ZnS, 8 per cent., with a freezing-point at approximately 1,045°C. Zinc sulphide is practically insoluble in lead sulphide when solid, and probably *vice versâ*.

Zinc sulphide-silver sulphide form a eutectic of the composition ZnS, 3, and Ag_2S, 97 per cent., with a freezing-point at 800°C. ZnS is practically insoluble in Ag_2S. Ag_2S is probably practically insoluble in ZnS, or only so to a very limited extent. This is of interest again in connection with the condition of the precious metals in mattes, already discussed.

II. Description of the Results Obtained from the Chemical and Microscopic Examination of Various Samples of Mattes.

Specimens of matte made by different methods of smelting were procured from widely-scattered plants in the United States and samples prepared for microscopic and chemical examination. In order to have a close correspondence between the chemical composition of a sample and its microscopic section, the piece for the section was roughly cut and then the grindings obtained in preparing for the polished face were taken for the chemical analysis. It was found that the sample obtained in this way, while at times somewhat contaminated with material from the emery-wheel, was by far the best for the purpose, since it afforded certainty that the chemical analysis corresponded to the microscopic section. No thermal analyses of mattes were attempted at this time, as the probable complexity of the cooling-curves and their proper interpretation promised no definite results. (See Section III. of this paper.)

After the examination of many sections of mattes and in

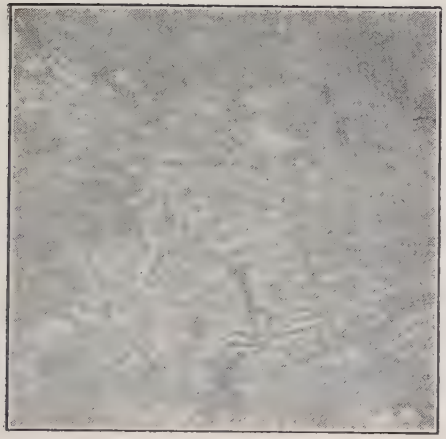
consideration of the facts established by the results on the various binary systems outlined in Section I., we believe the following constituents to be present in mattes. The constituents were identified by their color, hardness (relief polish), and their behavior under etching, as recommended by the authors of the various researches on the binary systems of the sulphides, who worked chiefly with nearly pure material. In general, it may be said that etching yields on the whole but indifferent results in the case of mattes, and that the best results are obtained by the judgment of relative color under constant light, and relief polish, though etching is an aid to be employed in certain instances.

Constituents of Copper-Iron and Copper-Lead-Iron Mattes.

1. *A Substance Hereafter Called "D."*—Under direct-reflected daylight it varies from a gray-blue to a sky-blue to a purple-blue color. With high-power incandescent electric light the blue color is somewhat lighter. This material may be either pure cuprous sulphide, Cu_2S, or Cu_2S + Cu, passing from this to a mixed crystal of Cu_2S-Cu holding in solution FeS-Fe to a probable extent of 25 per cent. of FeS-Fe in 75 per cent. of Cu_2S-Cu. When nearly pure Cu_2S, it always shows the characteristic crystal boundary-lines of that substance; its color is then gray-blue to sky-blue. As the amount of iron increases the color passes to darker blue and purplish blue.

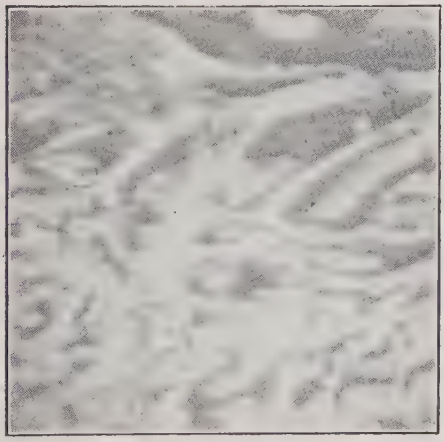
In sections of mattes high in copper, showing large fields of this material, if rapidly cooled, thus giving a fine-grained structure, a cloudy yellowish effect is often obtained, which is either an incipient eutectic, "*B*" (mentioned below), or the precipitation of FeS-Fe, from a saturated solution on cooling. The substance "*D*" is rarely porous, and usually presents a rather smooth uniform surface. In relief polish it shows itself to be the softest of the various constituents usually present in mattes. It frequently contains within itself in seams a substance hereafter called "*F*," of metallic luster and a copper color, which tests show to be pure or nearly pure metallic copper. This substance is further described below.

2. *A Substance Hereafter Called "C."*—Under direct-reflected daylight it shows usually a gray brass yellowish color. Hofman calls this a "fawn color." With incandescent electric illumina-



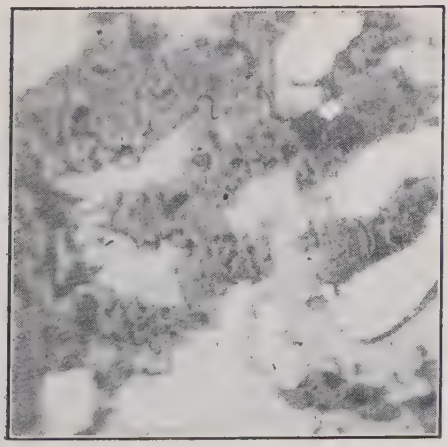
(Magnified 121 diameters.) Typical eutectic "*B*" and some excess substance "*C*." Light portion of eutectic is "*D*;" darker portion, "*C*." Light spot in lower right-hand corner is "*A*." Darkest spots are cavities.

FIG. 14.—COPPER-IRON MATTE. STANDARD SMELTING CO. No. 1*a*.



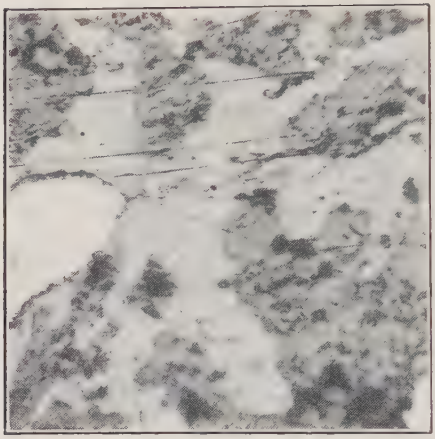
(Magnified 385 diameters.) Eutectic "*B*" under high magnification. Light portion is substance "*D*;" dark portion is "*C*." Field shows the tendency to coalescence.

FIG. 15.—COPPER-IRON MATTE. STANDARD SMELTING CO. No. 1*b*.



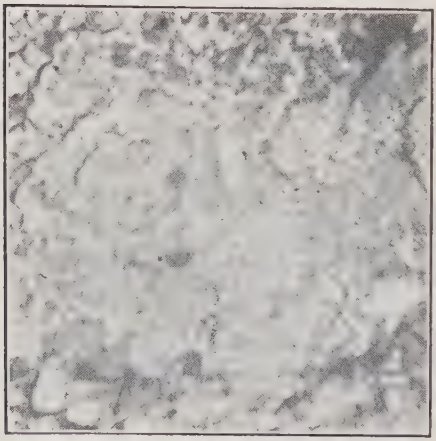
(Magnified 385 diameters.) The lightest constituent of the field is "*A*." Where this occurs in areas of "*D*," the next darker substance, it is bordered by "*F*." The black spots are cavities, which occur in areas of "*C*." Coalescence has obliterated eutectic structure.

FIG. 16.—COPPER-IRON MATTE. STANDARD SMELTING CO. No. 3.



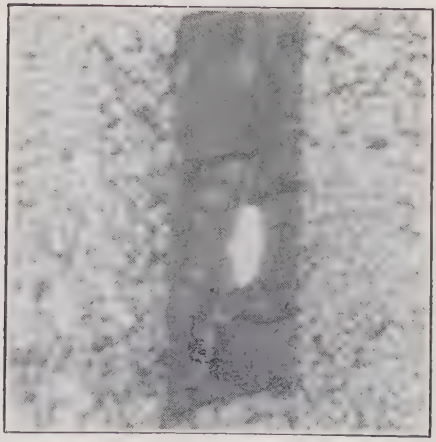
(Magnified 385 diameters.) White mass to the left is "*A*" (metallic iron), bordered by "*F*." Field is evenly divided between "*D*" (light portion) and "*C*" (darker portion). Darkest spots are cavities. Lines in the field are scratches.

FIG. 17.—COPPER-IRON MATTE. STANDARD SMELTING CO. No. 5.



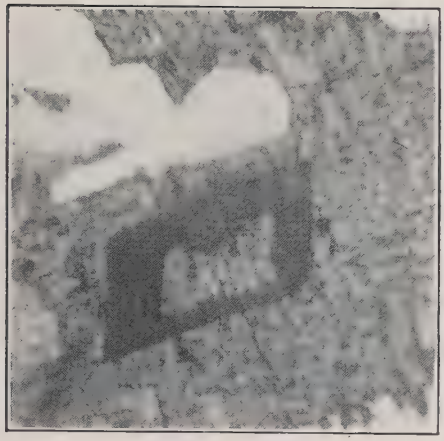
(Magnified 87 diameters.) Relief-polished field. The white areas standing most in relief are "*A*." To the right is some eutectic "*B*," which is also scattered through the rest of the field in small areas. The darkest spots are cavities. Some rhomboid crystals of "*G*."

FIG. 18.—COPPER-IRON MATTE. STANDARD SMELTING CO. No. 10*a*.



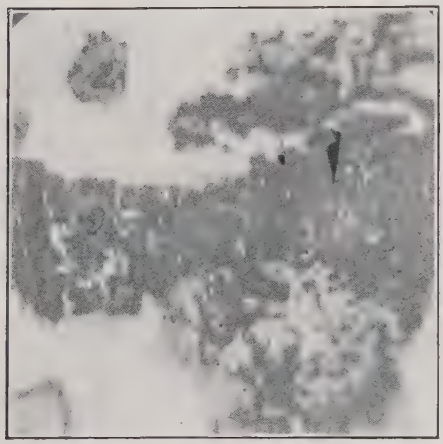
(Magnified 385 diameters.) A crystal of "*G*" in eutectic "*B*." The eutectic is poorly developed. The white area in the center of the crystal is "*A*."

FIG. 19.—COPPER-IRON MATTE. STANDARD SMELTING CO. No. 10*b*.



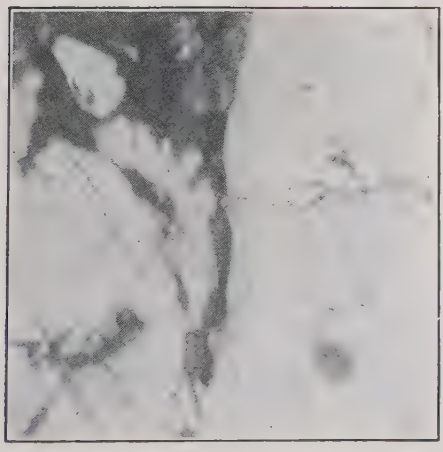
(Magnified 385 diameters.) Crystal in center is substance "*G*." White mass is "*A*." Balance of field is eutectic "*B*" poorly developed.

FIG. 20.—COPPER-IRON MATTE. STANDARD SMELTING CO. No. 10*c*.



(Magnified 385 diameters.) The dark area through the middle is a seam of "*F*." The lightest portion of the field is "*D*." The somewhat darker portion is "*C*," with dark spots, which are cavities.

FIG. 21.—COPPER-IRON MATTE. MAMMOTH COPPER CO. No. 21*a*.

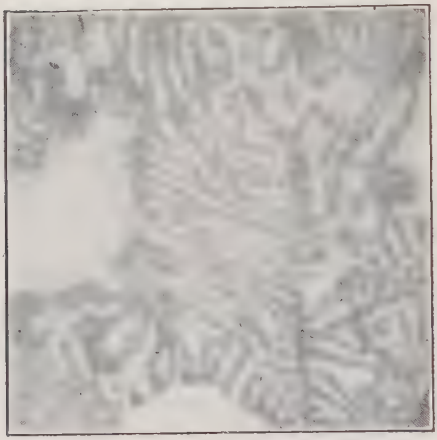


(Magnified 385 diameters.) The light portion of the field is substance "*D*." The dark portion is "*F*" (metallic copper).

FIG. 22.—COPPER-IRON MATTE. MAMMOTH COPPER CO. No. 21*b*.

(Magnified 385 diameters.) Field divided between eutectic "*K*" and substance "*D*" as excess substance. Dark places in latter are seams of "*F*" (metallic copper).

FIG. 23. — COPPER-LEAD-IRON MATTE. DURANGO, COLO. No. 26*a*.



(Magnified 385 diameters.) The field shows eutectic "*K*" and areas of "excess" substance "*D*." The dark portion of the eutectic is substance "*L*," which in this instance was tarnished, thus taking dark. Refer to Fig. 25.

FIG. 24. — COPPER-LEAD-IRON MATTE. DURANGO, COLO. MATTE No. 26*b*.

(Magnified 385 diameters.) Center of field shows eutectic "*K*," the light parts of which or "islands" are substance "*L*." This takes light in this instance, as in the specimen it was untarnished. The light uniform substance in the field is "*D*." The darker area at the bottom is a field of "*C*," inclosing some "*D*" and some cavities. Some "*F*" also shows, in the area of "*D*" at the left.

FIG. 25. — COPPER-LEAD-IRON MATTE. ARGO PLANT. REVERBERATORY MATTE No. 28.

tion a lightening of the color is observable. In mattes very low in copper it sometimes has a rosin color like that of very pure blende. This material is either pure iron sulphide, FeS, or more usually FeS dissolving some Fe (2 per cent.), or, in the presence of Cu_2S, is a mixed crystal of FeS-Fe dissolving as a maximum approximately 5 per cent. of Cu_2S. Cavities and blow-holes are of frequent occurrence in this material. This substance readily takes on a tarnish when exposed to the air, assuming a purplish tinge.

3. *A Conglomerate Substance—a Eutectic—Hereafter Called "B."*—Its individual constituents are the substances "*D*" and "*C*." In some of the mattes examined this eutectic is very clearly developed. As already mentioned, the eutectic composition between copper and iron sulphide occurs at what corresponds to about Cu_2S, 21, and FeS, 79 per cent., so that mattes low in copper show the existence of eutectic very much better than those high in copper. The eutectic in the mattes examined was both fine-textured and coarse-textured, dependent upon the rate of cooling. Some of the mattes examined consisted almost entirely of eutectic. Many of the specimens show a very strong tendency to the coalescence of the individual constituents of the eutectic, so that the phenomena of eutectic and large masses of both free mixed crystals "*D*" and "*C*" can be found in an apparent violation of the phase rule. This coalescence of the individual constituents of the eutectic is more marked as the amount of eutectic decreases, *i. e.*, the percentage of copper increases, and in mattes of from 45 to 60 per cent. of copper the substance "*C*" frequently occurs in isolated masses thrown out by the coalescence or segregation of the substance "*D*" (of the eutectic) with the great mass of excess substance "*D*." In very rapidly cooled mattes it takes powers of from 400 to 900 diameters to resolve the eutectic into its constituents, but in slowly-cooled mattes low powers show it plainly. Specimens on both sides of the eutectic composition were examined, and eutectic and excess substance clearly identified.

4. *A Substance Hereafter Called "A."*—Under direct-reflected daylight and electric light it has a brilliant white color. It is usually present in irregularly rounded and elongated grains, often of considerable size, evidently of roughly spherical

or ellipsoid form. This substance answers to all tests for metallic iron. It is not a common constituent of mattes; in fact, it is usually absent in ordinary matte. However, we had at our disposal numerous samples of matte produced in conjunction with a highly siliceous slag, at a high furnace-temperature, and were able to study thoroughly this rather rare constituent of matte. In the specimen examined it was usually surrounded by eutectic "*B*" or the mixed crystals "*C*" and "*D*." Wherever it was in contact with "*D*" seemed to be a favorite place for the appearance of seams filled with "*F*," or metallic copper. In relief polish "*A*" showed itself to be the third hardest substance present, coming after "*C*." It was found in considerable quantities in some of the matte specimens mentioned above, constituting from one-third of the bulk downward.

5. *A Substance Hereafter Called* "*F*."—In direct-reflected daylight and electric light it shows a copper color, and answers to all tests for metallic copper. It most frequently fills long, thin seams and occurs as minute crystals in cavities. Practically, it invariably occurs in fields of "*D*," and may be found in mattes of any copper-tenor from the lowest to the highest, though more of it occurs in mattes high in copper, as will be noted later. It is the so-called "moss copper" of mattes, and is discussed more in detail later in Section III. of this paper.

6. *A Substance Hereafter Called* "*G*."—In direct-reflected daylight or electric light it shows a light-purple color. It is generally in the form of crystals of rhomboidal cross-section, sometimes very much elongated. It is translucent. Some of the crystals have zonal inclusions. At times, instead of crystals irregular grains of this substance are found. Its perfect crystal shape proclaims it to be the first of the constituents of the matte to take form. In most mattes but little of this substance is present, and in a number of specimens it is practically absent. It could not be positively identified, but evidence points it out to be zinc sulphide. It occurs most plentifully in mattes containing considerable zinc. The mixed crystal "*C*" is probably capable of dissolving about 3 per cent. of ZnS.

7. *A Substance Hereafter Called* "*L*."—In direct-reflected daylight and electric light it shows a light-gray color. It has a uniform peculiarly smooth appearance. Tests show it to be

practically pure lead sulphide, PbS. It is found only in mattes containing lead in appreciable quantities.

8. *A Conglomerate Substance—a Eutectic—Hereafter Called "K."*—Its individual constituents are "*L*" (lead sulphide) and "*D*" (mixed crystal, cuprous sulphide-iron sulphide), already described in detail. The eutectic is usually rather fine in texture, very delicate and smooth in appearance, and of typical structure. It shows the same tendency to segregation that the eutectic "*B*" does. The eutectic between lead sulphide-copper sulphide has been discussed under the binary systems.

9. *A Substance Hereafter Called "M."*—By direct-reflected daylight and electric light it has a light green-gray color. It appears usually very sparingly as long, thin strings of crystal grains. Its positive identification was not possible, but we think it to be included slag-crystals.

The above seven substances and the two eutectics comprise all the constituents of copper-iron and copper-iron-lead mattes that we have found. Table IV. gives a description of the samples examined, the constituents being designated by their respective letters; in many instances the chemical analysis also is given.

Summary.

1. *Iron-Copper Mattes.*—All iron-copper mattes examined show themselves decidedly to be conglomerates, as indicated by Bolles.[28] There is a distinct eutectic present, often very beautifully developed, especially in slowly-cooled mattes. This eutectic is best defined in mattes of low copper-content, as already mentioned. In mattes of 35 per cent. of Cu and above, it will, in most cases, be faint or absent, due to the coalescence of the individual constituents of the eutectic. In a number of specimens examined this coalescence could be plainly traced, some specimens showing on the one side of the section perfectly developed fine eutectic structure, which gradually passed to a coarse eutectic structure, and from that to the two individual constituents of the matte, "*C*" and "*D*," in isolated fields. When the matte contains about 60 per cent. of copper, or 75 per cent. Cu_2S-Cu, and 25 per cent. FeS-Fe, no more "*C*" is visible, as 25 per cent. of FeS-Fe is the amount of this

[28] *Trans.*, xxxv., 666 (1905).

TABLE IV.—*Description of Copper-Iron and Copper-Iron-Lead Mattes.*

Number.	Name of Plant Sample is Derived from.	Microscopic Examination. Contains. "A."	"B."	"C."	"D."	"F."	"G."	"L."	"K."	"M."	Chemical Composition. Cu. Per Ct.	Fe. Per Ct.	S. Per Ct.	Pb. Per Ct.	Zn. Per Ct.	Remarks.
1	Standard S. Co. Rapid City, S.D.	Considerable.	Considerable.		Much as "excess."	Some.										Partial pyrite-smelting; highly siliceous slag.
2		Some.	Very much.													
3		Some.	Much.		Some as "excess."						23.63	64.22				
4		Very little.	Considerable.	Considerable as "excess."												
5		Some.	Much.	Considerable as "excess."												
6		Very little.	Much.	Some as "excess."			Some.									
7		None.	Much.		Some as "excess."	Some.	Some.									
8		Some.	Much.		Some as "excess."	Some.					18.38	62.45				
9		Considerable.	Much.	Some as "excess."							13.89	60.54				
10		Much.	Considerable.		Some as "excess."	Some.	Some.									
11		Some.	Considerable.	Some as "excess."												Typical specimen.
12		Considerable.	Little.		Much as "excess."	Considerable.	Some.									
13		Considerable.	Much.	Much as "excess."												Partial pyrite-smelting; highly siliceous slag.

											Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	
14	.....	None.	Much.	.....	A little as "excess."	Considerable.	.....	.....	.....	.....	24.43	42.76	.....	.....	.....	
15	..... Standard S. Co. Rapid City.	Some.	Much.	Considerable as "excess."	.....	.....	.....	.....	.....	.....	12.33	51.60	.....	.....	.....	Partial pyrite-smelting; highly siliceous slag.
16	.....	Considerable.	Much.	Some as "excess."	.....	Some.	.....	.....	.....	.....	14.08	56.54	.....	.....	.....	
17	.....	None.	Much.	.....	Some as "excess."	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	
18	.....	None.	Much.	.....	Some as "excess."	.....	.....	.....	.....	.....	21.64	43.70	.....	.....	.....	
19	Mammoth Copper Co., Kennett, Cal.	None.	Much.	.....	Some as "excess."	.....	.....	.....	.....	.....	19.95	43.35	.....	.....	.....	
20	.....	None.	Much.	.....	Some as "excess."	.....	.....	.....	.....	.....	22.69	41.46	.....	.....	.....	Pyrite-smelting; 3½ per cent. coke.
21	.....	None.	Very little.	Segregated fields.	Much as "excess."	Considerable.	.....	.....	.....	.....	50.2	19.80	.....	.....	.....	
22	Standard Smelting Co., Rapid City, S. D.	Very little.	Very much.	.....	.....	.....	.....	.....	.....	.....	19.94	50.85	25.63	.....	.....	Partial pyrite-smelting; highly siliceous slag.
23	.....	None.	None.	None.	Considerable as "excess."	Some.	.....	Some as segregated fields.	Considerable.	Some.	46.55	7.06	.....	19.87	.....	From re-smelting of lead-copper mattes. Western plants.
24	American S. & R. Co., Omaha, Neb.	None.	.....	.....	Large uniform fields.	.....	.....	Some in round spots.	.....	Some.	74.26	1.41	16.24	3.26	.....	White metal from converting lead-copper mattes.
25	.....	None.	None.	Segregated fields.	Considerable as "excess."	Some.	.....	.....	Some.	.....	22.68	22.93	19.15	20.5	.....	Copper-lead matte; blast-furnace.
26	Durango, Colo....	None.	None.	None.	Considerable as "excess."	Considerable.	Some.	.....	Considerable.	.....	55.45	6.25	18.42	18.34	.....	Copper-lead matte; blast-furnace.
27	A. S. & R. Co., Murray, Utah...	None.	None.	.....	Considerable as "excess."	Some.	Some.	.....	Much.	.....	45.46	11.70	18.55	23.23	.....	Lead-copper matte; blast-furnace.
28	Argo Plant, Denver, Colo.....	None.	A little.	Considerable as segregated fields.	Much as "excess."	Some.	Some.	.....	Little.	.....	41.51	22.47	24.75	4.91	3.43	Reverberatory matte.

TABLE IV.—*Description of Copper-Iron and Copper-Iron-Lead Mattes.—Continued.*

Number.	Name of Plant Sample is Derived from.	Microscopic Examination. Contains. "A."	"B."	"C."	"D."	"F."	"G."	"L."	"K."	"M."	Chemical Composition. Cu.	Fe.	S.	Pb.	Zn.	Remarks.
											Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	
29	Perth Amboy, N. J.	None.	Some.	Some as segregated fields.	Some as "excess."				Some.		35.46	21.97	20.11	11.41		Blast-furnace copper-lead matte. Cerro de Pasco ores.
30		None.	Much.	Some as "excess."	Small segregated fields.						10.49	54.66	26.27			Pyrite-smelting.
31	Tennessee Copper Co., Tennessee.	None.	Considerable.		Some as "excess."	Some.	Considerable.				23.96	42.89	26.39			Pyrite smelting.
32		None.	Some.	Considerable as segregated fields.	Much as "excess."	Considerable.					41.10	29.45	24.23			Pyrite-smelting.
33	Montana Ore Purchasing Co., Butte, Mont.	None.			Large fields.	Considerable.					71.56	5.10	23.32			Converter white metal.
34	Copper Queen Cons. M. Co., Bisbee, Ariz.	None.			Large fields.	Some.					80.84	0.55	19.60			Converter white metal.
35		None.	Some.	Considerable as segregated fields.	Much as "excess."	Some.					40.25	29.78	24.91			Blast-furnace matte.
36	Mountain Copper Co., Pt. Lewis, Cal.	None.	Some.	Considerable as segregated fields.	Considerable as "excess."	Very little.	Considerable.				36.48	32.66	25.20		3.05	Oil-fired reverberatory matte.
37		None.	Some.	Considerable as segregated fields.	Considerable as "excess."	Little.	Some.				45.10	26.63	24.54			
38		None.	Some.	Considerable as segregated fields.	Considerable as "excess."	Some.	Some.				45.00	27.21	25.30			
39		None.	Some.	Considerable as segregated fields.	Considerable as "excess."	Some.	Some.				44.94	25.35	24.99			

substance soluble in Cu_2S-Cu, forming the saturated mixed crystal "*D*." In mattes approximating this composition, but just below it, in fact down to 40 per cent. of copper, eutectic is practically never found, due to the strong coalescence above mentioned, and the substance "*C*" is present in isolated fields within substance "*D*."

A number of specimens contained many cavities and cracks, these being mainly in substance "*C*." Cavities are but rarely found in substance "*D*." When the composition of the matte approaches that of white metal, substance "*D*" shows well-developed cleavage- and crystal-boundary planes. Inclusions, like gas-bubbles, are frequently present. The substance "*F*," or metallic copper, is of common occurrence, and may be found in mattes of almost any copper-tenor. It most generally occurs as thin, flat sheets and wires, which, in the microscopic section, appear as long, thin seams. These invariably occur in substance "*D*," so that this must be looked upon as the source of the metallic copper. This substance "*F*" (metallic copper) usually increases in quantity with the copper-content of the matte. It is present in considerable quantity in many mattes which do not show its presence to the naked eye. The substance "*G*," or zinc sulphide, is a common constituent, although usually present in small amounts. Its general form is that of rhomboidal, translucent, purplish crystals, well developed, though it may be present as irregular grains in some very rapidly cooled mattes. The constituent "*A*," metallic iron, is present only in mattes produced under certain conditions, which have been discussed.

2. *Copper-Iron-Lead Mattes.*—In general appearance the structure of these mattes is similar to that of the copper-iron mattes, except that the eutectic usually present is "*K*," that of Cu_2S and PbS. In some cases both eutectics, "*B*" (Cu_2S-Cu, FeS-Fe) and "*K*" (PbS, Cu_2S-FeS), are found. The presence of lead sulphide, in commercial lead-copper mattes, usually reduces the amount of FeS, so that, in many instances, all of this substance is in solution in the Cu_2S-Cu.

The eutectic "*K*" shows the same tendency of the coalescence of its individual constituents, "*D*" and "*L*," that the eutectic "*B*" does, though in most mattes where it is not present in great amount there is no opportunity for this coalescence, as it is the last of all the constituents to freeze.

The reason for the structure of mattes is developed in Section III., which follows. The microphotographs accompanying this paper were selected chiefly with a view to illustrate structure. They are fully described in the notes.

III. The Constitution of Mattes.

It is desirable to append a discussion of the theoretic constitution of the most complex of the mattes under examination, *i. e.*, the iron-copper-lead mattes, in order to explain the results obtained in the microscopic examination. The above matte should be considered as belonging to a ternary system, the members of which are assumed to be Cu_2S, PbS, and FeS, though in reality the end members will be Cu_2S-Cu, PbS, and FeS-Fe, as outlined in the first part of this paper. In our opinion there is no reason for viewing this as a quaternary system, Cu-Fe-Pb-S, since the sulphide compounds actually exist at the temperatures under discussion. From the literature available to us it seems to be most generally assumed that in a ternary system the freezing-point of the ternary eutectic lies below that of any of the eutectics of the binary systems entering into the ternary system. It can, however, be mathematically demonstrated, graphically and analytically, by an application of Raoult and van't Hoff's law, that the ternary eutectic freezing-point may lie either below, at the same height (temperature ordinate), or above the lowest of the binary eutectics. In general, it is apt to lie above the lowest binary eutectic, when the freezing-points of the eutectics of the other two binary systems are relatively much higher than that of the third or lowest. Raoult and van't Hoff's law of the depression of the freezing-point is applicable only to substances insoluble in each other in the solid state. It applies practically to two of the binary systems—PbS-FeS and PbS-Cu_2S—and not to Cu_2S-FeS. The graphical demonstration of a ternary system can well be made according to the equilateral-prism[29] method, the freezing-point curves of the binary systems being plotted on the sides of the prism, ordinates indicating temperatures. In the case of simple V curves, the point of intersection of the three surfaces passed through each two branches of the V curves common to one constituent

[29] Findlay, *The Phase Rule*, pp. 231, 245.

of the ternary system marks the position of the ternary eutectic, as regards both composition and temperature of freezing.

Three binary systems, PbS-Cu_2S, FeS-PbS, and FeS-Cu_2S, enter into the ternary system under discussion. These have been fully discussed in the first part of this paper, but it is desirable to repeat the composition and freezing-points of their eutectics. Cu_2S-FeS: eutectic composition, Cu_2S, 21; FeS, 79 per cent.; freezing-point, 895°C. Cu_2S-PbS: eutectic composition, Cu_2S, 51; PbS, 49 per cent.; freezing-point, 540°C. PbS-FeS: eutectic composition, FeS, 25.8; PbS, 74.2 per cent.; freezing-point of eutectic, 782°C.

From a graphical demonstration, taking into account the nature of the Cu_2S-FeS curve, it is evident that the ternary eutectic point lies above that of the lowest binary eutectic—namely, PbS-Cu_2S at 540°C. Therefore the ternary eutectic is a "hidden eutectic," and it follows from analysis that no true ternary eutectic, *i. e.*, a conglomerate of the three substances, FeS, PbS, and Cu_2S, can exist in the solid mass. In no instance did we find the slightest indication of a ternary eutectic in the mattes examined, but very decided cases of two binary eutectics. If the ternary eutectic freezing-point is lower than that of any of the three binary eutectics a ternary eutectic conglomerate should be found. From analysis it would also appear that the main eutectic to be found in a case such as the above is that of the lowest freezing-point, *i. e.*, PbS-Cu_2S, or "*K*." This was invariably present in the lead-copper-iron mattes examined.

In order to demonstrate the constitution more clearly we have outlined below the freezing of copper-lead-iron matte, making certain assumptions as follows:

1. That the composition of the "hidden" ternary eutectic is PbS, 40; FeS, 25; and Cu_2S, 35 per cent., with a freezing-point at 600°C.

2. For the sake of simplicity in calculation the eutectic composition of the binary eutectics is assumed as follows:

a. Cu_2S-FeS containing Cu_2S, 20; FeS, 80 per cent., or taking into account that 80 per cent. of Cu_2S dissolves 20 per cent. of FeS, a eutectic composition of FeS, 75; Cu_2S-FeS, 25 per cent., with a freezing-point at 895°C.

b. PbS-FeS containing PbS, 72, and FeS, 28 per cent., freezing at 782°C.

c. Cu_2S-PbS containing Cu_2S-FeS, 50, and PbS, 50 per cent., freezing at 535°C., this again taking into account the solubility of FeS in Cu_2S.

Case 1. Taking 100 g. of matte of the following composition: FeS, 25; PbS, 25; and Cu_2S, 50 per cent., and recalculating in order to provide for the FeS dissolved in the Cu_2S, we have 62.5 g. of Cu_2S-FeS, 12.5 g. of FeS, and 25 g. of PbS. Above 600°C., the ternary eutectic point, the excess substance is all that portion of the matte in excess of the ternary eutectic composition. The ternary eutectic composition of this matte, according to the percentage-assumption above, is 25 g. of PbS, 27.3 g. of Cu_2S-FeS, and 10.1 g. of FeS, a total of 62.4 g. The excess substance is therefore 35.2 g. of Cu_2S-FeS and 2.4 g. of FeS, a total of 37.6 g. As the matte cools towards 895°C., the point at which the eutectic between Cu_2S and FeS forms, the first true excess, Cu_2S-FeS, freezes out, and when 895°C. is reached 34.4 g. of this substance, or substance "*D*," will have separated out. Then at 895°C. the eutectic between Cu_2S and FeS freezes out, the amount being 3.2 g., composed of 2.4 g. of FeS and 0.8 g. of Cu_2S-FeS.

The mother-metal is now of ternary eutectic composition and simply cools from 895°C. to 600°C., remaining molten during this interval. Since a binary eutectic between PbS and Cu_2S exists which has a lower freezing-point (535°C.) than that of the "hidden" ternary eutectic, no true ternary eutectic conglomerate can freeze out at 600°C., but at this temperature the "excess" substance over the lowest binary eutectic will separate out. This excess substance, according to calculation, is 10.1 g. of FeS and 2.3 g. of Cu_2S-FeS, and is in reality in surfusion, and while, according to calculation, it represents an excess of 3.2 g. of FeS, and 9.2 g. of eutectic "*B*" (6.9 g. of FeS and 2.3 g. of Cu_2S-FeS), it probably freezes out at once, showing only incipient eutectic structure. The mother-metal now has the eutectic composition of the eutectic between PbS and Cu_2S-FeS and cools down to 535°C., when it freezes to the eutectic "*K*," or Cu_2S-FeS, 50, and PbS, 50 per cent., or, in actual amount, 50 g. composed of 25 g. each of the above-named substances, or "*D*" and "*L*."

It is to be noted that this manner of freezing, with the separating-out of the eutectic "*B*" at higher temperature while a

large mass of mother-metal is molten, gives the greatest possible opportunity for the coalescence of the individual constituents of this eutectic, so that it is rather to be expected that segregated fields of the individual substances rather than eutectic structure should be found. This was the actual result of the microscopic examination of the copper-lead-iron mattes. The eutectic "*B*" was found, but only in isolated instances.

Case 2. Taking 100 g. of matte of the following composition: Cu_2S, 18; PbS, 20; and FeS, 62 per cent., and recalculating to provide for the FeS dissolved in the Cu_2S, we have Cu_2S-FeS, 22.5; PbS, 20; and FeS, 57.5 per cent. The ternary eutectic of this matte according to the percentage-composition is 20 g. of PbS, 21.8 g. of Cu_2S-FeS, and 8.1 g. of FeS. The excess substance is, therefore, 0.7 g. of Cu_2S-FeS and 49.4 g. of FeS. As the matte cools towards 895°C., FeS separates out as the first true excess, and when this temperature has been reached 47.3 g. of this substance, or "*C*," will have separated out. Then the eutectic "*B*" freezes, amounting to 2.8 g., composed of 0.7 g. of "*D*" and 2.1 g. of "*C*." The mother-metal is now of ternary eutectic composition and cools to 600°C., when the excess substance, 8.1 g. of FeS and 1.8 g. of Cu_2S-FeS, separates out, as described in Case 1. The remaining mother-metal now cools and finally freezes to 40 g. of eutectic "*K*," as described for Case 1.

Case 3. Taking 100 g. of matte of the following composition: PbS, 70; FeS, 20; and Cu_2S, 10 per cent. (this is of unusual composition as regards ordinary lead-copper-iron mattes, and is taken here only to outline the method of freezing), and recalculating to provide for the FeS dissolved in the Cu_2S, we have PbS, 70; Cu_2S-FeS, 12.5, and FeS, 17.5 per cent. The ternary eutectic of this matte according to the percentage eutectic composition is 12.5 g. of Cu_2S-FeS, 11.4 g. of PbS, and 4.6 g. of FeS. The excess substance is therefore 58.6 g. of PbS and 12.9 g. of FeS, but it must be borne in mind that in order to form the final binary eutectic, "*K*," 12.5 g. of PbS are required, therefore the real amount of the excess is 57.5 g. of PbS and 12.9 g. of FeS.

As the matte cools towards 782° C. (the point at which the eutectic between FeS and PbS forms), PbS separates out as

excess substance, until at 782° C. 24.3 g. has separated out; then the eutectic of these two substances freezes, amounting to 46.1 g., and composed of 33.2 g. of PbS and 12.9 g. of FeS. The mother-metal then simply cools to 600° C., the ternary eutectic point, when excess substance, FeS, freezes out, amounting to 4.6 g. The remaining mother-metal, then of eutectic "*K*" composition, cools to 535° C., when it freezes as the eutectic "*K*," amounting to 25 g. The composition of the matte as frozen then is:

First excess, PbS, or "*L*,"	= 24.3 g.	
First binary eutectic, PbS-FeS,	PbS = 33.2 g.	46.1 g.
	FeS = 12.9 g.	
Second excess, FeS, or "*C*,"	= 4.6 g.	
Second binary eutectic,	Cu_2S-FeS = 12.5 g.	25 g.
Cu_2S-FeS-PbS, or "*K*,"	PbS = 12.5 g.	
Total, ...	100.0 g.	

These three cases illustrate the manner of freezing of a copper-lead-iron matte, and tend to explain the constitution as found by microscopic analysis. The freezing of an ordinary copper-iron matte is much simpler and need not be further discussed. There seems to be reason to believe that in the case of copper-iron matte containing metallic iron the method of freezing is a similar one, as is indicated by the absence of any evidence of ternary conglomerate eutectic, but the presence of fields of metallic iron, indicating it as the first excess substance to be frozen out. The case of the probable zinc sulphide crystals, substance "*G*," is explained in the same manner. If the iron and zinc sulphide do enter into any eutectic during freezing, the very powerful tendency toward coalescence, already discussed, explains their presence as isolated constituents.

The Relation of the Microscopic Constitution to the Chemical Composition of Mattes.

It has been frequently noted that the sulphur-content of mattes does not correspond with that called for by considering the copper, iron, lead, etc., combined as the sulphides, Cu_2S, PbS, and FeS. Table V. gives the composition of mattes based on the theory that the metals are present as sulphides only.

TABLE V.—*Calculated Composition of Mattes Having the Metals Present as Sulphides.*

Copper.	Iron.	Sulphur.	Total.
Per Cent.	Per Cent.	Per Cent.	Per Cent.
00.0	63.64	36.36	100 = FeS
7.99	57.28	34.73	100
11.985	54.094	33.92	100
31.96	38.184	29.856	100
47.94	25.456	26.604	100
63.92	12.728	23.352	100
79.9	0.00	20.1	100 = Cu_2S

The range of sulphur in true mattes, including lead-mattes, is from 18 to 27 per cent. A very common figure for ordinary copper-iron mattes is in the neighborhood of 23 or 24 per cent. The sulphur is lowest in copper-iron-lead mattes and highest in copper-iron mattes containing much iron. It is plain that the sulphur-content is below that called for by the assumption that all of the metal present is combined as the sulphide. From what has gone before it is not necessary to give further explanation. Considering the temperature of the production of most mattes, it is evident that the end members are not Cu_2S, FeS, PbS, etc., but are Cu_2S + Cu, FeS + Fe, PbS + Pb. This is also borne out fully by the microscopic examination. Witness the almost invariable presence of metallic copper, and in certain rare instances the large amount of metallic iron. In ordinary mattes the metallic iron is not visible microscopically, being in solution in the FeS and possibly in the Cu_2S-FeS to some extent. When present, however, in any quantity, it shows up very plainly. Some further discussion is necessary in reference to the metallic copper.

According to the researches of Heyn and Bauer[30] and of other observers, metallic copper is insoluble in cuprous sulphide in the solid state. The work of the above-named authors did not include the dimorphic point of Cu_2S, which occurs at 103°C. We have frequently observed at the plant of the Standard Smelting Co. the presence of metallic copper, so-called "moss copper," in large quantities, comparatively speaking, in mattes of a low copper-tenor, as low as 10 per cent. of copper. In watching the cooling of casts of this matte it was

[30] *Metallurgie*, vol. iii., No. 3, p. 84 (Feb. 8, 1906).

noticeable that this "moss copper" appeared suddenly when the matte was nearly cold, relatively speaking, but still too hot to bear the hand upon it. Our experience is strikingly confirmed by Palmer[31] and also by E. L. Larison.[32]

In view of these facts we suggest tentatively that the dimorphic point, 103°C., marks the throwing-out of metallic copper from solution in the Cu_2S-FeS. It has already been shown that this metallic copper originates exclusively in fields of Cu_2S-FeS, or substance "*D*," and not in FeS, or "*C*." The presence of "moss copper" in mattes of such low copper-content will seem unusual to some metallurgists, but the explanation readily follows when it is considered that these mattes were produced by partial pyrite-smelting with a slag containing from 46 to 49 per cent. of SiO_2, necessitating a high temperature. The presence of metallic iron in considerable quantity in these mattes indicates the temperature employed. Ordinarily "moss copper" visible to the naked eye is not seen in mattes containing less than 30 per cent. of copper.

Mattes as a Carrier of the Precious Metals.

As to which constituents of a matte act as solvents for the precious metals has already been discussed in Section I. of this paper. To recapitulate: Cuprous sulphide and silver sulphide form an unbroken series of mixed crystals, and there is little doubt that when cuprous sulphide is present silver is in solution in this substance as sulphide. Metallic copper, usually present in copper-mattes, has a powerful solvent action on both silver and gold. Cuprous sulphide has a powerful solvent action on gold. Lead sulphide has a considerable solvent action on silver sulphide, but probably only a limited one on gold. Zinc sulphide has very little solvent action on silver sulphide and probably little on gold. Metallic iron has a powerful solvent action on gold and but a very limited one on silver. Ferrous sulphide has practically no solvent action on silver sulphide or on gold. Arsenic and antimony are present usually in very small amounts, and in view of the well-known feeble solvent action of "speisses" we believe that their presence has practically no significance as a

[31] *Mining and Scientific Press*, vol. xciii., No. 20, p. 604 (Nov. 17, 1906).
[32] *Mining World*, vol. xxvii., No. 14, p. 550 (Oct. 5, 1907).

solvent for the precious metals. The substances in mattes, therefore, which dissolve precious metals and give the matte value as a "collector" are cuprous sulphide, metallic copper, lead sulphide, and metallic iron. In our opinion, in order to collect both gold and silver satisfactorily, cuprous sulphide, or metallic copper, which implies the first, is essential.

IV. Conclusions.

From the foregoing it appears that mattes in structure are a conglomerate, and including copper-iron and copper-iron-lead mattes they contain the following common constituents:

1. A substance called "*D*," of Cu_2S dissolving as a maximum 33 per cent. of its weight of FeS, or 75 per cent. of Cu_2S dissolves 25 per cent. of FeS. This substance is dissociated at the temperatures of matte-formation, so that it finally becomes of a composition which may be written $(3\ Cu_2S, 1\ FeS) - S = (x\ Cu_2S, y\ FeS + w\ Cu + z\ Fe)$, the amount of metallic iron and metallic copper present being a function of the temperature. The metallic copper is thrown out of solution at 103°C.

2. A substance called "*C*," of FeS dissolving as a maximum approximately 5.2 per cent. of Cu_2S, or 95 per cent. FeS dissolves 5 per cent. of Cu_2S. This, as the above, is dissociated by the temperature of matte-smelting until it becomes of the composition $(95\ FeS, 5\ Cu_2S) - S = (x\ FeS, y\ Cu_2S + w\ Fe + z\ Cu)$.

3. A conglomerate called "*B*," being a mixture of the two substances "*D*" and "*C*," occurring at a composition of about 17 per cent. of Cu, and freezing at 895°C. when no dissociation has taken place.

4. A substance called "*A*," consisting practically of metallic iron.

5. A substance called "*F*," consisting practically of metallic copper.

6. A substance called "*G*," probably nearly pure zinc sulphide.

7. A substance called "*M*," probably slag-crystals.

8. A substance called "*L*," consisting practically of pure PbS, lead sulphide.

9. A conglomerate called "*K*," consisting of a mixture of the substances "*D*" and "*L*" (definite proportions of eutectic

not known), nearly pure PbS and Cu_2S, forming a eutectic at Cu_2S, 51; and PbS, 49 per cent., and freezing at 540°C.

In conclusion, we desire to thank all those who have assisted us with samples of matte and in other ways have aided the work represented by this paper.

Particular thanks are due to Prof. M. F. Coolbaugh and D. C. Woodward in connection with chemical analyses, and to Oscar Anderson for drawing the curves accompanying this paper.

Professional Ethics.

BY JOHN HAYS HAMMOND, NEW YORK, N. Y.

(Chattanooga Meeting, October, 1908.)

THIS is an era of "expansion;" and, conformably with the change in commercial conditions, the function of the mining engineer, as well as that of his *confrères* in many other professions, has also expanded. From the position of an engineer, limited to the determination of technical questions, the engineer of to-day has come to assume an economic importance in those branches of industry dependent upon engineering skill for their development.

He is indeed an engineer of limited usefulness who does not go further professionally than to submit a purely technical report on subjects presented for his consideration. While he has the same responsibility as formerly in the solution of the technical problems involved, he is further expected to supplement his report with advice on the financial and commercial aspects of these problems. For the great majority of problems presented to the engineer ultimately involve the determination of the pecuniary relations of the propositions under consideration.

The great corporation lawyer no longer earns his fee by merely submitting his opinion as to the legal issues involved in law-suits affecting the corporation; he now assumes charge of the entire legal work, including both the organization of the corporation and the maintenance of its legal status thereafter. In like manner, the engineer, whose report was, as I

have said, formerly confined to the determination of the geological and other cognate features of a mining enterprise, is now expected, not only to embody information on these subjects in his report, but also to give his opinion whether the property offered for sale in pursuance of such an enterprise is worth the price asked for it.

Moreover, if he recommends the purchase of a property, he incurs a certain moral responsibility for its efficient management, inasmuch as his professional reputation depends upon the realization of his predictions as to the outcome of the investment. For this reason the engineer passes from the rôle of an expert to that of a consulting engineer, bound to supervise both the technical and the business management of the property purchased by his clients.

With this additional responsibility, and the consequent increase of professional opportunity, comes an extension of the field covered by the ethics of the mining engineer. Our professional duty may be regarded from three points of view: first, the relation of our work to our employers; second, its relation to the investing public; and third, its relation to our individual interests.

Relation to Employers.—The question has been often discussed among engineers whether it is professionally proper to make a report for the seller of a mining property. Such practice has been sometimes condemned; but a little reflection will show that the condemnation is not warranted. For it is neither wrong nor unreasonable that the owner of a mineral property should desire to present such a statement of its nature and probable value as will secure the attention of possible purchasers; or that he should secure for such a purpose the assistance of one who knows how to make such a statement intelligently and in scientific language—*i.e.*, the language which will convey a definite meaning to those who know it—or that he should pay such an assistant for his skill and labor. It goes without saying that no expert in such a case, or in any other case, may suppress or misstate facts, or disguise or withhold his honest opinion, in a statement to which he lends his name. Yet, this being understood, it still remains true that a "vendor's report" is not advice to anybody to buy, at a stated price, but only a description of the property (usually without any mention

of the price at which it would be to the purchaser a profitable bargain). There is nothing dishonorable or unprofessional in honestly making, for the owner of the property, such a description. But there is often danger that it may be afterwards used as if it were advice offered to a purchaser. In any such report, therefore, the fact that it was made for the owner or vendor, or his representatives, should be clearly stated beyond all misunderstanding, for the protection of both the author and the reader. This precaution being observed, I see no valid objection to the writing of such a report by a mining engineer.

Nevertheless, it is an undesirable kind of work, particularly for young mining engineers, whose reputation for integrity and judgment, not yet established by long practice, may be, justly or unjustly, placed in jeopardy by any such expression of their opinions, and may be destroyed, in public esteem, by the failure of any mining scheme with which their names may have been thus associated. In their own interest, they should accept such engagements with caution, and protect themselves with vigilance against illegitimate or misleading use of their statements. In short, they take, in such cases, a risk which older engineers, backed by a reputation already established, could better afford to take. As between the two classes, the question is one, not so much of ethics as of practical wisdom; yet we may fairly say that the avoidance of folly is, in no small degree, an ethical matter. It is worthy of notice that the wise proverbs and warnings of the Good Book often make no great distinction between "fools" and "sinners."

In this connection arises the question of the acceptance of contingent fees. It seems to me that an expert of established and unimpeachable reputation is perfectly justified in recommending the purchase of a property and accepting compensation from the promoters, provided, first and without qualification, that full publicity be given to the nature of his connection with the enterprise; and secondly, that his compensation be contingent, not upon the successful sale of the property, but upon the subsequent success of the undertaking. In other words, an expert who makes a favorable report upon a mining property, for which he is to receive payment in money if the property be sold on his report, must needs have an unparalleled reputation for integrity to endure that revelation of this situa-

tion which he cannot without dishonor withhold. But one who frankly avows that he is to be paid in the stock of the projected company, and therefore, as an investor of time, skill, and labor, is on the same footing as the investors of money, occupies a much less vulnerable position. In either case, and in any case, there is no safety—and, I may almost add, no honor—outside of the frank and full publication of all such circumstances as might be afterwards called in question, criticised, or condemned.

This is the expert's Golden Rule: "Tell unto others, now, whatsoever you would not have them tell on you (with unjust misunderstanding and scandalous comment) hereafter!"

Another question involved in the relations of a mining engineer to his employer concerns the acceptance of commissions from the sellers of machinery, supplies, etc., the purchase of which he has made as agent or recommended as adviser. In many instances of similar nature affecting other professions, this practice seems to be established and condoned, if not avowedly justified. It is even defended by ingenious sophistry somewhat as follows: The manufacturers of certain articles are bound by trade-agreements not to sell them below a certain price, or, in the absence of such agreements, do not dare to "cut" the price, for fear of thereby invoking savage reprisals from their competitors. But this minimum price includes the salary or commission paid to a regular selling-agent. If, therefore, the agent of a customer makes a purchase direct, without the intervention of their sales-agent, they are willing to pay him "the regular commission," though they are not willing to reduce by the same amount their bill to his employer. The argument is plausible from the manufacturer's standpoint; and indeed it is hard to judge him justly. For if, in a given case, in order to gain a customer, he undersells his competitors, he may be committing the heinous modern sin of granting a "rebate;" whereas, if he refuses to do this, he may be guilty of partnership in a wicked "trust" or "combine." The popular and legislative *furore* which condemns at the same time free competition in prices and every agreement to prevent such competition, needs, as President Roosevelt has, in substance, frankly declared, to be modified by common sense.

But we are not now concerned with the ethical or legal problems of the manufacturer. From the standpoint of the engi-

neer, acting as a purchasing-agent or adviser, the case seems to me as clear in the court of honor as it would be in a court of law. He cannot honorably accept a commission from the seller while he is the agent of the buyer. If the custom of the trade permits the giving of such a commission, but not its deduction from the face of the bill rendered, he may, of course, honorably accept it and pay it over to his own employer—that being the only way in which he could secure the minimum net price in his employer's interest. But even in that event he should place his honor beyond suspicion—as, for instance, by demanding the commission in a check to his own order, and indorsing the same check to the order of his employer. There may be exceptional cases in which the taking and keeping of a commission is justifiable; but one thing is clear beyond dispute: it is always wrong when it needs to be kept secret.

It seems almost superfluous to insist upon the duty of maintaining a spirit of loyalty towards employers. Yet this spirit is by no means as prevalent as a sense of honor, or even of enlightened self-interest, would demand. Loyalty to a corporation should be as unquestionable as to an individual. In neither case should it be sacrificed to feelings of personal resentment or disapproval. If one cannot "stand for" the policies or practices of one's employer, the straightforward course is to seek employment elsewhere.

There are two special spheres in which the mining engineer, like the members of other technical professions, maintains a double relation.

One of these is that in which, being himself an employee, he exercises the employer's authority over other employees. In such a position, while it is his duty to protect vigilantly his employer's rights and interests, he should realize that he cannot do this more effectively than by invariable justice, good-nature, respect, and sympathy towards his own subordinates. Nothing contributes more to the success of an enterprise than the mutual exercise of these feelings between workmen and superintendents or managers. It is "team-work" that wins in industrial undertakings, as well as in the athletic field. And in these days especially it is the cultivation of an *esprit de corps*, based upon truly friendly personal relations, which offers the

best protection against the schemes of mischief-makers and demagogues.

The other double relation to which I have alluded is that which is occupied by an expert witness in a court of law. In this case, he is theoretically *amicus curiæ*, a friend of the court, giving under oath his assistance in the recognition and interpretation of the facts of the case; and, for this reason, he is permitted to do what no ordinary witness can do—namely, to give opinions as well as personally observed facts, and to support these opinions by hearsay evidence or scientific authority and argument, not otherwise admissible as testimony. Yet, on the other hand, he is retained and paid by one of the litigant parties; and he would not appear in court at all were it not reasonably certain beforehand that his testimony would support the theory and benefit the cause of his clients.

The ethics of this situation may be more clearly perceived if we consider that, first and foremost, the expert witness must remember the obligation of his oath. He has not the same latitude as the attorney, whose official duty is to present a client's case, without declaring his private opinion about it. Consequently, the expert witness should be thoroughly convinced beforehand of the justice and truth of the cause or the theory in support of which he appears. Such a conviction ought to be based upon a careful and exhaustive previous examination of the case; and no expert should agree to testify in any case until such an examination has established his opinion and qualified him to defend it.

Concerning some aspects of this question, I would here refer, without further discussion in this place, to the suggestive address of Dr. Raymond, printed in the *Proceedings of the American Institute of Electrical Engineers* for November, 1906.

Relations to the Public.—A more simple and congenial, yet more heavily responsible, relation is that which the mining engineer may be called on to assume towards the public, as the responsible indorser of a proposed undertaking. In consideration of the confidence which he invokes and expects, notwithstanding the fact that he has been paid by interested parties, he assumes a sacred trust, obligating him to safeguard the interests of a wider and more important clientage. The investor has not the time, and perhaps not the technical knowledge,

required for the thorough study of a prospectus. He risks his money upon his belief in the ability and integrity of the engineer, who is therefore bound in prudence and in honor to be specially careful that his indorsement be clear and precise, without mental reservation or opportunity for misunderstanding. This caution applies particularly to what are called "gilt-edge" investments—that is, to enterprises so promising and so thoroughly investigated beforehand as to warrant their recommendation, on the terms stated, to the general public—that is, to unknown and probably unskilled clients. There are numerous instances in which the engineer has to give advice on speculative, though honest and legitimate, undertakings, such as the development of mining prospects, which should be confined to those who are willing, and can afford, to take greater risks, in the hope of larger profits, than pertain to the "gilt-edge" class of mining investments.

Relations to Personal Interest.—In connection with the class of advisory reports last named, it is good policy for the expert adviser to seek to acquire a special clientage among mining investors, who will not be led to judge him by the results of a single case, but, knowing the wisdom and the successful outcome of his recommendations in other cases, will not condemn him when he turns out to have been mistaken. For such a disappointment, sooner or later, is inevitable, since no engineer, however able and careful, is infallible.

In my opinion, a mining engineer engaged in such consulting practice should aim to become himself a capitalist, and a purchaser and developer of mining properties. Should he acquire business experience, he might become avowedly a promoter, in a high and worthy sense of that term, and thus assist in driving out the unscrupulous promoters who are really parasites. According to my observation, the swindling in mining enterprises is done mostly by laymen, and not by responsible professional engineers or practical miners. The practical miner, indeed, is, in many instances, himself the victim of the unscrupulous promoter.

As already observed, in considering the relations of the engineer to the public, it may also be repeated, from the standpoint of his own interests, that he should use the greatest circumspection to prevent the use of his name in support of enterprises

of which he has but a superficial knowledge. He may be quite willing to "take the chances," and invest some of his own money in a scheme which he would not be willing to take the responsibility of recommending to another party, dependent upon his judgment as a guide. But even such a private investment of his own money should often be avoided, lest his connection with the scheme as a simple stockholder should be construed as implying his deliberate indorsement of it as an expert.

It often happens that an expert, who has won deserved reputation and public confidence, is called to give advice concerning a district with which he is not personally acquainted. In such a case, his honest off-hand judgment, based upon general knowledge and upon experience elsewhere, might mislead both himself and others. Yet it is practically impossible for an expert, visiting a given property or district for the first time, and, of course, known to all as the possible "advance agent" of capital, to get at the unfavorable conditions which it is everybody's interest to hide. I would earnestly recommend, in such cases, the engagement by the examining expert, at his own selection and cost, and without notice to any other party, of a local expert assistant, familiar with the conditions and history of the property concerned, and of the district in which it lies—in other words, with those facts which a visitor cannot easily discover.

I am fully aware that these desultory remarks have covered but a small part of the field of professional ethics, even in a single branch of the professions represented by the Institute. But I am not offering a code, or even an official and representative declaration of principles. My remarks are intended, and will, I hope, be accepted, simply as a contribution to that discussion of their theme in which every engineer of experience may participate.

The Behavior of Calcium Sulphate at Elevated Temperatures with Some Fluxes.

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(Chattanooga Meeting, October, 1908.)

I. Introduction.

The mineral gypsum, $CaSO_4 + 2\ H_2O$, has been used for many years as a sulphurizing and basic flux in several smelting-operations. Thus, in smelting oxide nickel-ore in the blast-furnace, it is commonly added to the charge to furnish the sulphur necessary for collecting the metal in a matte, and a base for slagging the siliceous gangue. In the concentration of lead-copper matte in the reverberatory furnace it has been used for years at Freiberg, Saxony,[1] for a similar purpose, and for producing at the same time a copper-matte with less than 0.15 per cent. of Fe. The latest use gypsum has been put to is in the blast-roasting process of Carmichael-Bradford.[2] The term "blast-roasting," given by A. S. Dwight[3] to the Dwight-Lloyd method of roasting and agglomerating,[4] is a happy generic term which covers the ground better than the "lime-roasting" of Ingalls[5] or the "pot-roasting" of Austin,[6] in that it leaves the operation independent of the character of the flux and the form of apparatus, and retains the characteristic feature of this class of processes—namely, that of using forced draft.

In the Carmichael-Bradford process the dehydrated material, mixed with galena-concentrate, acts as a diluent and a flux. The process differs in this from the Huntington-Heberlein[7] and the Savelsberg[8] processes, in both of which limestone is used.

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[1] Hofman, *Metallurgy of Lead*, 2d ed., p. 372 (1899).
[2] *U. S. Patent No.* 705,904 (July 29, 1902).
[3] *Engineering and Mining Journal*, vol. lxxxv., No. 13, p. 649 (Mar. 28, 1908).
[4] *U. S. Patents Nos.* 882,517 and 882,518 (March 17, 1908).
[5] *Engineering and Mining Journal*, vol. lxxx., No. 9, p. 402 (Sept. 2, 1905).
[6] *Mining and Scientific Press*, vol. xciii., No. 17, p. 511 (Oct. 27, 1906).
[7] *U. S. Patent No.* 600,347 (March 8, 1898).
[8] *U. S. Patent No.* 755,598 (March 22, 1904).

While there has been some speculation[9] as to the behavior of calcium sulphate in blast-roasting, there is no direct evidence as to what causes its decomposition in the operation. The results obtained in practice with the Carmichael-Bradford process at Port Pirie, N. S. W.,[10] show that the gases issuing from the converting-pots are rich enough in sulphur dioxide (as high as 10 per cent. of SO_2 by volume) to be utilized in the manufacture of sulphuric acid, while with the Huntington-Heberlein and the Savelsberg processes the percentage of sulphur dioxide in the gases is too low to make them available for this purpose.

The following calculation proves that the high content of sulphur dioxide in the gases of the Carmichael-Bradford process must be due to the decomposition of calcium sulphate along certain lines. Pure galena contains, S, 13.4 per cent. The charge of the Huntington-Heberlein process, as worked in Europe, contains, Pb, about 50 per cent., and S, from 13 to 14 per cent. Of the total sulphur, 6.7 per cent. comes from the galena, the rest from the associated metallic sulphides. Pure gypsum contains, S, 23.54 per cent. In making up a charge with 20 per cent. of gypsum, the flux will add 4.7 per cent. of S to the mixture, or, with the charge above, nearly 76 per cent. of the sulphur-content of the galena. With a charge containing as little as 20 per cent. of lead, the sulphur-addition through the gypsum will be twice as large as that through the galena. If it can be proved that gypsum is decomposed in a manner to furnish sulphur dioxide, the high percentage of this gas in the products passing off from the Carmichael-Bradford converters will have been fully explained.

The object of the present investigation was to study the behavior of gypsum when exposed to elevated temperatures, both alone and in the presence of fluxes found in mixtures made up for blast-roasting sulphide lead-ores. The leading

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[9] *Engineering and Mining Journal*, Clark, vol. lxxviii., No. 16, p. 630 (Oct. 20, 1904); No. 18, p. 708 (Nov. 3, 1904); Hutchings, vol. lxxx., No. 16, p. 726 (Oct. 21, 1905); Editor, vol. lxxxii., No. 19, p. 883 (Nov. 10, 1906); Anstee, vol. lxxxii., No. 21, p. 984 (Nov. 24, 1906); Hutchings, vol. lxxxiii., No. 4, p. 201 (Jan. 26, 1907); *Metallurgie*, Borchers, vol. ii., No. 1, p. 1 (Jan. 8, 1905); *Chemiker Zeitung*, Weiller, vol. xxxii., No. 27, p. 342 (Apr. 1, 1908); Tandler, vol. xxxii., No. 35, p. 437 (Apr. 29, 1908).

[10] Clark, *Engineering and Mining Journal*, vol. lxxviii., No. 18, p. 708 (Nov. 3, 1904).

fluxes are silica, iron oxide, and lead oxide. The results obtained also throw some light upon the general behavior of calcium sulphate in smelting.

II. Preparation of Raw Materials.

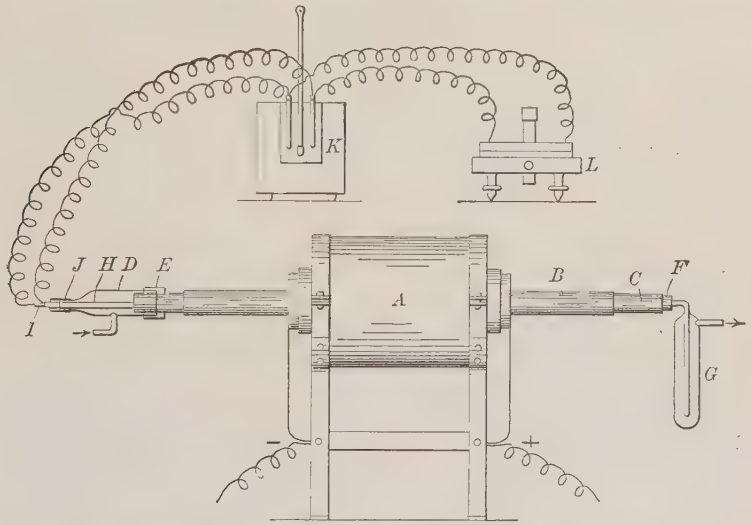
In carrying on experiments in the dry way with substances in quantities of 0.5 g. or less, it is essential that these be chemically pure, as the presence of even a slight quantity of impurity not only disturbs accurate observation, but may cause complications through side-reactions. A striking example of this is furnished by the observations of Doeltz and Mostowitsch,[11] who found, in determining the melting-point of barium sulphate, that the chemically pure salt of Kahlbaum, containing, however, traces of iron, melted at 1,453° C., while the melting-point of the salt, when absolutely chemically pure, was found to lie at 1,580°. While it was difficult to determine quantitatively the trace of iron, its presence was shown by a reddish tinge in the impure salt after fusion.

1. *Preparation of Calcium Sulphate.*—The start was made with chemically pure calcium carbonate of Kahlbaum and ammonium sulphate of Merck. The calcium carbonate gave, upon analysis, $CaCO_3$, 99.2 per cent., and showed the presence of MgO and Fe_xO_y; the ammonium sulphate gave, upon ignition, a residue of only 0.025 per cent. The calcium carbonate was treated as follows: It was dissolved in hydrochloric acid, and ammonia added to the incipient precipitation of iron hydroxide; from the calcium chloride solution, free from iron, the lime was precipitated twice as oxalate to free it from magnesia; the calcium oxalate was then ignited to constant weight. The calcium oxide, thus purified, was dissolved in hydrochloric acid, neutralized with ammonia, and a small excess of ammonium sulphate added. The precipitated calcium sulphate was washed until free from ammonium sulphate and dried. The dehydrated salt, upon analysis, gave CaO, 41.23, while theory calls for 41.20 per cent.

2. *Preparation of Silica.*—Previous experience had shown that it was next to impossible to obtain from ground silica, sold as chemically pure, a product with less than from 0.3 to 0.4 per cent. of impurity, which usually is composed of magnesia, lime,

[11] *Zeitschrift für anorganische Chemie*, vol. liv., No. 1, p. 146 (May 28, 1907).

and alumina in the form of silicate or aluminate. The impurities are either present in the raw material, or have been introduced by the grinding-apparatus. The preparation of pure silica by precipitating with sulphuric acid, electrolyzing the colloidal precipitate in connection with dialysis, takes much time and does not furnish a product of better grade than is obtained by using a colorless transparent quartz-crystal. A specimen of Brazilian quartz, obtained from the Mineralogical Department of the Massachusetts Institute of Technology, was brought to



A. Heraeus electric-resistance tube-furnace. *B.* Heating-tube, of unglazed Marquardt material. *C.* Quartz tube for platinum boat. *D.* Glass adapter-like connection with air-inlet. *E.* Rubber connection. *F.* Rubber stopper. *G.* Railroad-tube. *H.* Quartz tube for thermo-electric wires. *I.* Marquardt tube for isolating the thermo-electric wires from each other. *J.* Rubber connection. *K.* Cold-junction cast-iron block. *L.* Siemens-Halske millivoltmeter.

Fig. 1.—Heating, Measuring, and Testing Apparatus.

red heat, quenched in water, and pulverized in a diamond mortar; coarse particles of steel were removed with a magnet, and the fine powder treated repeatedly in a platinum dish with hydrochloric acid until the last trace of iron disappeared, then washed with water, dried and ground fine in an agate mortar. Two samples of 2 g. each gave with hydrofluoric acid a residue of 0.0007 and 0.0008 g., showing that the powder contained SiO_2, 99.96 per cent. In treating silica from a quartz with hydrofluoric acid, it is necessary to warm the powder for

several hours with repeated additions of the acid to volatilize all the silica.

3. *Preparation of Lead Oxide.*—The simplest way of obtaining pure lead oxide is to ignite in an electric furnace at from 800° to 850° C. lead peroxide deposited electrolytically from a solution of chemically pure lead nitrate, strongly acidified with nitric acid. This method of preparation was followed.

4. *Preparation of Ferric Oxide.*—This was prepared from Mohr's salt, $(NH_4)_2SO_4,FeSO_4 + 6H_2O$. This salt, usually very pure, was further purified by re-crystallization; it showed upon analysis, Fe, 14.22 per cent., the theoretical composition calling for 14.25 per cent. The salt was first heated over a Bunsen burner, then transferred to a porcelain boat and held at 1,000° C. in an electric furnace to constant weight.

III. Heating, Measuring, and Testing Apparatus.

The general arrangement of the apparatus used is shown in Fig. 2. The air, before entering the furnace at the left, is purified by passing through wash-bottles and a drying-tower; the right end of the heating-tube is covered with an annealing-cup, which is replaced by a railroad-tube with indicator solution, as shown by *G* in Fig. 1, if the issuing gases are to be tested; the electric current, from the 110-volt circuit shown in the upper left-hand corner of Fig. 2, passes through three rheostats on the floor and an ammeter on the table before it enters the furnace and returns to the switch; the cast-iron cold-junction block and the millivoltmeter for the thermo-electric couple are seen on the table to the left of the furnace.

The furnace, *A*, Fig. 1, used for heating, was an electric-resistance tube-furnace of W. C. Heraeus, Hanau, Germany. Its heating-tube, *B*, of unglazed Marquardt material, was 44 cm. long, had an inner diameter of 20 cm., and was wound for a length of 20 cm. with platinum foil of 0.007 mm. thickness. With the highest permissible current density of 10 amperes at 110 volts, the maximum temperature attainable was 1,250° C. An attempt to use a stronger current caused the platinum foil to burn out. After removing it, the tube was re-wound with 100 cm. of platinum foil, 6 mm. wide and 0.05 mm. thick, leaving spaces of from 2 to 2.5 mm. between the windings. The ends of the foil were joined to platinum wires, 1 mm. thick, and the latter connected with the poles of the furnace. With the new winding,

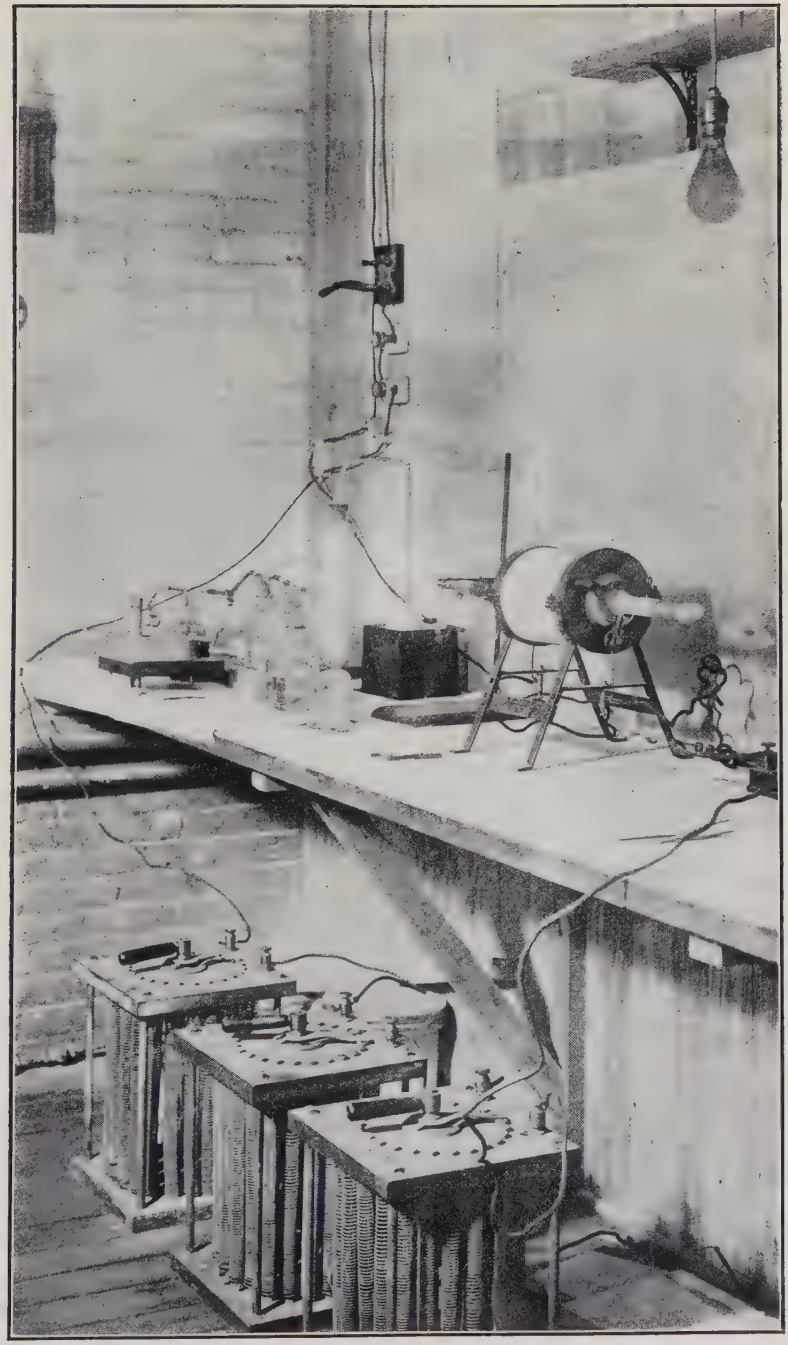


FIG. 2.—ARRANGEMENT OF APPARATUS.

FIG. 4.—CRYSTAL-FORMS OF CALCIUM FERRITE.

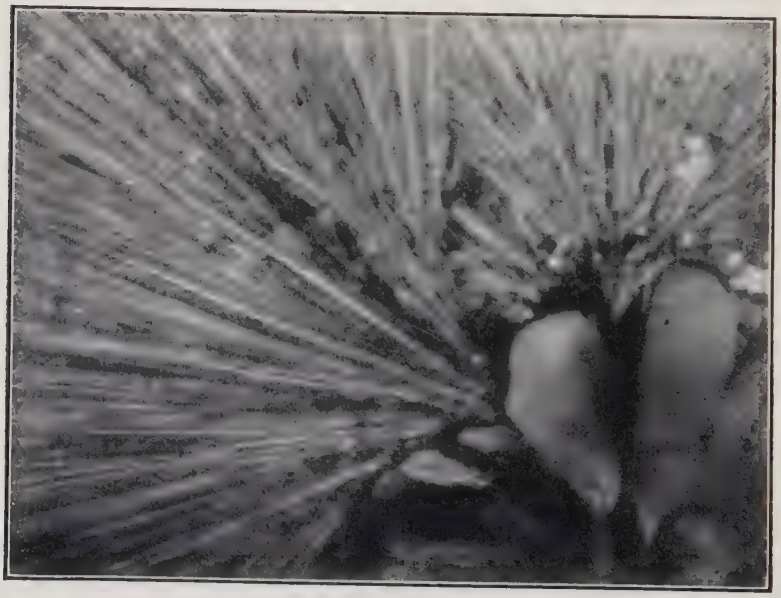


FIG. 5.—CRYSTAL-FORMS OF CALCIUM FERRITE.

a temperature of 1,500° C. was easily obtained, but a stronger current and a longer time were required for heating up the furnaces than with the old winding. Thus, in starting with a current of 12 amperes, the density soon falls on account of the growing resistance of the heating-coil; cutting out resistance to maintain the current approximately constant at from 9 to 9.5 amperes, a temperature of 1,000° will be attained in from 1 to 1.5 hr. Above 1,000°, the temperature can be raised more quickly. With the 110-volt circuit, in periods of from 25 to 30 min. the temperature was brought from 1,000° to 1,200° with 13.5 amperes, to 1,300° with 14.5 amperes, to 1,400° with 16.5 amperes, and to 1,500° with 18 amperes.

Inside the Marquardt tube, *B* of Fig. 1, was placed the quartz tube, *C*, 60 cm. long and 14 mm. inner diameter. It served its purpose well, as it is infusible at 1,500° C., remains gas-tight at this temperature, and can be easily cleaned after a test by treating with suitable reagents, followed by washing and heating. This mode of procedure was followed after every test. A further advantage of the quartz tube lies in its low heat-conductivity, which permits the use of rubber connections at the ends. Thus, with 1,400° at the center, the temperature at the ends does not exceed 50°.

The platinum boat used for holding the charges was 5 cm. long by 1 cm. wide by 0.8 cm. deep. The charges were spread to a thin layer at the end in contact with the junction of the thermo-couple.

Temperatures were measured with a Le Chatelier thermo-electric pyrometer with cast-iron cold-junction block, *K*, and a Siemens-Halske millivoltmeter, *L*, having 180 decimillivolt-divisions for 1,700° C. The wires were incased in a quartz tube, of 4 mm. inner and 6 mm. outer diameter, closed at one end by fusing with an oxy-hydrogen gas blow-pipe; they were isolated from each other by the Marquardt tube, *I;* a rubber tube, *J*, made an air-tight joint between the adapter-like glass connection, *D*, and the quartz tube, *H*. Air was forced through the glass tube by means of a Beutell blast-apparatus connected with the water service-pipe of the laboratory. With a very small consumption of water, a pressure of 50 cm. of water was always available. By the use of screw-clamps the velocity of the air-current was easily regulated.

All the experiments were carried on in air that had been purified and dried. For this purpose the air was passed first through a $\frac{1}{10}$ normal solution of potassium permanganate to oxidize any organic matter and anything that might have a reducing effect, then through two wash-bottles containing respectively concentrated solutions of potassium and barium hydroxide to absorb all carbon dioxide—as long as the solution of barium hydroxide remained clear, the potassium hydroxide was not neutralized; the third wash-bottle, containing concentrated sulphuric acid, dried the air and delivered it to a drying-tower, 26 cm. high and 5.5 cm. in diameter, the lower half of which was filled with soda-lime, the upper half with granular calcium chloride.

For the testing of the products of decomposition, railroad-tubes, *G*, were charged with a colorless solution of potassium iodate and starch[12] for the detection of sulphur dioxide; with barium chloride (1:20) acidulated with hydrochloric acid for sulphur trioxide; with barium chloride (1:20) acidulated with hydrochloric acid and charged with bromine for both sulphur dioxide and trioxide. The last solution is very sensitive, as it becomes decidedly cloudy with a few tenths of a milligram of sulphur dioxide.

IV. Behavior of Calcium Sulphate at Elevated Temperatures.

The statements of the leading chemical manuals regarding the behavior of calcium sulphate at elevated temperatures are indefinite and contradictory. O. Dammer[13] asserts that calcium sulphate fuses at a red heat without being decomposed, and solidifies as an anhydrous salt. R. Abegg[14] declares that, according to Mitscherlich and Zulkowsky, calcium sulphate is decomposed at a white heat into calcium oxide and sulphur trioxide. H. Moissan[15] says that calcium sulphate fuses at a dark-red heat without decomposition, that at a white heat it is split into calcium oxide and sulphur trioxide, and that 0.5 g.

[12] Hofman, *Trans.*, xxxv., 817 (1905).

[13] *Handbuch der anorganischen Chemie*, Enke, Stuttgart, vol. ii., part 2, p. 315 (1894).

[14] *Handbuch der anorganischen Chemie*, Hirzel, Leipsic, vol. ii., part 2, p. 136 (1905).

[15] *Traité de Chimie Minérale*, Masson & Co., Paris, vol. iii., p. 550 (1904).

of the salt heated before the blow-pipe for 20 min. is completely converted into oxide. A. Ditte[16] says that at the melting-temperature of iron ("*fer*") calcium and barium sulphates gradually give up their acid, which is split into sulphur dioxide and oxygen, while the respective oxides remain behind as residues. H. F. Morley and M. M. P. Muir,[17] H. E. Roscoe and C. Schorlemmer[18] have little to say upon the subject.

In studying the decomposition of $CaSO_4 + 2\ H_2O$, the first step was to find the temperature at which the salt gave up the last of its combined water. From 0.2 to 0.4 g. of the moist substance was heated in purified dry air in the platinum boat. The start was made at 500° C., and the heating prolonged every time to constant weight. After each heating the sample was placed in a desiccator containing sulphuric acid and soda-lime, kept there for a fixed time, and then weighed to an accuracy of ± 0.1 mg.; a number of samples tested in this way gave concordant results; therefore only a few need be recorded, which is done in Table I. At 1,180° the solutions in the absorption-tubes remained clear, proving that neither sulphur trioxide nor dioxide was given off. The table shows that $CaSO_4 + 2\ H_2O$ gives up all of its water at about 900°, and that between 900° and 1,800° it undergoes no further change.

TABLE I.—*Expulsion of Water from Hydrated Calcium Sulphate.*

Sample No.	Character of Substance.	Weight of Substance Before Heating.	Temperature.	Time.	Weight of Substance After Heating.	Loss in Weight.	Loss in Weight or Water.
		Grams.	Degrees C.	Min.	Grams.	Grams.	Per Cent.
1	Air-dry hydrated calcium sulphate containing 20.25 per cent. of water.	0.3577	500	60	0.2866	0.0711	19.87
			600	40	0.2863	0.0714	19.96
			700	40	0.2859	0.0718	20.07
			800	40	0.2855	0.0722	20.18
			900	60	0.2853	0.0724	20.24
			950	40	0.2853	0.0724	20.24
			1,000	30	0.2853	0.0724	20.24
			1,100	30	0.2853	0.0724	20.24
2	.....	0.4194	1,180	30	0.3344	0.0850	20.26

In all the following experiments at higher temperatures, the moist hydrated calcium sulphate used was heated to 1,150° C. to constant weight, and the weight thus obtained was taken as that of the anhydrous sulphate.

[16] *Etude générale des sels*, Dunod-Pinat, Paris, vol. ii., p. 325 (1906).

[17] Watt's *Dictionary of Chemistry* (1888). [18] *Treatise on Chemistry*, 3d ed. (1898).

The results obtained at temperatures above 1,150° are given in Table II. They show that calcium sulphate begins to become dissociated at 1,200° C., and that the loss in weight at a given temperature decreases with repeated heating. Sample No. 4, heated for 10 min. at 1,400°, was found to have melted to an enamel-like mass, and to have lost 30.2 per cent. in weight; continuing the fusion at 1,400° for 10 min. more increased the loss in weight to a total of 43.1 per cent. As calcium sulphate theoretically contains 58.79 per cent. of SO_3, the results prove that during fusion and even after fusion the dissociation is only in part; whether it is completed by prolonged fusion was not ascertained.

TABLE II.—*Decomposition of Anhydrous Calcium Sulphate by Heat.*

Sample No.	Character of Substance.	Weight of Substance Before Heating.	Temperature.	Time.	Weight of Substance After Heating.	Loss in Weight.	Loss in Weight.	Remarks.
		Grams.	Deg. C.	Min.	Grams.	Grams.	Per Cent.	
3	Anhydrous calcium sulphate.	0.252	1,200	15	0.2515	0.0005	0.28	$BaCl_2$-HCl-Br solution becomes turbid.
			1,200	15	0.2513	0.0002		
4		0.1973	1,200	15	0.1969	0.0004	0.30	
			1,200	15	0.1967	0.0002		
			1,300	10	0.1913	0.0006		
			1,300	10	0.1869	0.0044	7.7	
			1,300	10	0.1833	0.0036		
			1,300	10	0.1821	0.0012		
			1,400	10	0.1369	0.0604	30.2	Substance fuses.
			1,400	10	0.1112	0.0861	42.1	

In order to determine more accurately the melting-temperature of calcium sulphate, which has been shown to lie between 1,300° and 1,400°, a sample of 0.234 g. of anhydrous salt was brought to 1,300° and then heated for 3-min. periods in temperature-intervals of 10°. This narrowed the melting-range to 1,340° and 1,360°; at 1,340° the salt remained unchanged, at 1,360° it was fused and had suffered a loss in weight of 12.22 per cent. in the form of sulphur dioxide. A second sample, 0.199 g., treated in the same way, showed that the salt began to soften at 1,355° and was partly fused at 1,360°, accompanied by a loss in weight of 21.53 per cent. Boiling the second sample with water gave an alkaline solution (phenolphtalein) which contained both calcium oxide and sulphate. The question whether there is formed a basic salt or simply a mixture of calcium oxide and sulphate was not studied.

Summing up, the results show that anhydrous calcium sulphate treated in pure dry air at atmospheric pressure up to 1,200° C. remains unchanged; that at 1,200° dissociation begins, and increases with rising temperature; that at 1,360° fusion begins, accompanied by a part dissociation, the sulphur trioxide set free being split into sulphur dioxide and oxygen; that the composition of the product is governed by the temperature and time of heating; that therefore decomposition cannot be complete without fusion, as has been believed.

On account of the proven fact that the rate of decomposition of calcium sulphate by heat at and above fusion varies with the time given, it was necessary to adopt a definite mode of proce-

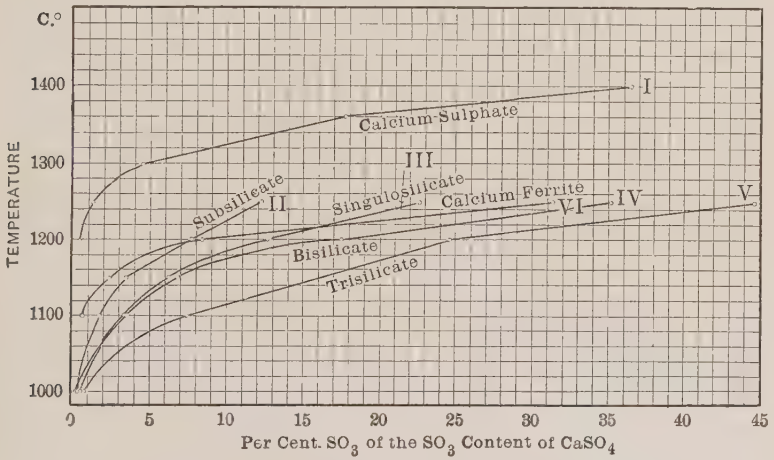


FIG. 3.—DECOMPOSITION OF CALCIUM SULPHATE.

dure, if results comparable with those from the use of fluxes were to be obtained. For this purpose a sample of 0.1694 g. of anhydrous salt was heated for periods of 10 min. in a temperature-range of from 1,200° to 1,400°. The results are given in Table III., and platted in Curve I. of Fig. 3.

All the curves in Fig. 3 represent heatings for 10-min. periods; the ordinates denote per cent. of SO_3 of the SO_3-content of $CaSO_4$; the abscissæ, degrees C.

The data obtained are similar to those already discussed. The curve brings out clearly how the dissociation, beginning at 1,200°, increases at an accelerating rate with the rise of temperature.

In passing, the similar behavior of barium sulphate[19] may be noted, which, at its melting-point of 1,580°, also undergoes a partial dissociation, losing 9 per cent. in weight, while the loss would have to be 34.29 per cent. if the decomposition had been complete.

V. Behavior of Anhydrous Calcium Sulphate with Silica at Elevated Temperatures.

Silica does not undergo any physical changes upon heating up to 1,600° C., when it begins to fuse, according to the researches of Day and Shepherd.[20] This figure replaces the higher values so far accepted.

In studying the effects of silica upon calcium sulphate, it was supposed that the decomposition of the latter would be accompanied by the formation of silicate, hence the mixtures to be heated in the electric furnace in a current of pure dry air were made up to correspond to definite silicates. The weighed quantities of calcium sulphate and silica were ground together in an agate mortar and the mixtures transferred as completely as possible to the platinum boat; this was placed in the furnace, held at 500° C., and the temperature then gradually raised. All charges assumed constant weight at 900°, the slight changes in weight experienced being due to the moisture absorbed in preparing the mixtures.

1. *Bi-Silicate Charge*, $CaSO_4 + SiO_2$.—Sample No. 6: At 1,000° C. the first product of decomposition was given off; this proved to be sulphur dioxide. Sample No. 7, 0.3733 g., heated to 1,300° for 60 min., experienced a loss in weight of 40.72 per cent.; the total SO_3 present was 40.52 per cent.; continuing the heating for 30 min. more gave no additional loss in weight; all the sulphur trioxide had been driven off and the decomposition was complete. Sample No. 8, 0.3444 g., heated to 1,250° for 60 min., lost 40.54 per cent. in weight instead of the theoretical 40.62 per cent.; further heating at 1,250° as well as at 1,300° showed no further diminution in weight; the decomposition at 1,250° was therefore also complete.

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[19] Doeltz-Mostowitsch, *Zeitschrift für anorganische Chemie*, vol. liv., No. 1, p. 148 (May 28, 1907).

[20] *Journal of the American Chemical Society*, vol. xxviii., No. 9, p. 1089 (Sept., 1906).

2. *Singulo-Silicate Charge*, $2\ CaSO_4 + SiO_2$.—Sample No. 9, 0.2346 g., which contained 23.10 per cent. of SiO_2 instead of the theoretical 22.18 per cent., gave off its first sulphur dioxide again at 1,000°; on heating for 60 min. to a temperature of from 1,250° to 1,300° it showed a loss in weight of 46.29 per cent. of SO_3 instead of the theoretical 45.23 per cent. (This difference as well as the others is explained by the impossibility of mixing the charge-constituents perfectly; thus one part of a charge will have contained a slight excess, another a slight lack of calcium sulphate.) As further heating gave no change in weight, the test shows that a singulo-silicate mixture is also decomposed completely.

3. *Tri-Silicate Charge*, $2\ CaSO_4 + 3\ SiO$.—Sample No. 10, 0.2430 g., just as the preceding silicates, gave off its first sulphur dioxide at 1,000°; on heating at 1,250° for 60 min. it showed a loss of 0.0857 g. SO_3 instead of the theoretical amount of 0.0858 g.; further heating gave no change in weight; the decomposition of calcium sulphate was therefore complete.

A microscopical examination of samples Nos. 7, 8, 9, and 10, with a magnification of 120 diameters, showed that they were all fritted and formed lumps, which presented a fern-like surface and had a strong luster, while the original mixtures appeared as a dull, loose powder, in which the calcium sulphate and the quartz-crystals could be readily distinguished. The changes indicate the formation of calcium bi-silicate, $CaSiO_3$; singulo-silicate, Ca_2SiO_4; and tri-silicate, $Ca_2Si_3O_8$. In order to prove this, the following experiments were made: From each silicate 0.1 g. was boiled with hot water, the solution filtered, and the filtrate tested for alkalinity and for dissolved lime. Phenolphtalein gave only a very slight alkaline reaction, and boiling with ammonia and ammonium oxalate a silky turbidity which was just visible. Both reactions prove that there was present only a very small amount of free lime. Further, the silicates were completely soluble in warm dilute hydrochloric acid; from these solutions ammonia precipitated silicic acid, and from the filtrates ammonium oxalate precipitated calcium oxalate. Parallel tests, with a corresponding amount of calcium oxide heated to 1,250°, showed that the solution obtained by boiling with water contained large amounts of lime, which showed a strongly alkaline reaction and gave a heavy precipi-

tate of oxalate. The behavior of carbon dioxide with the silicate and the lime gave similar evidence. The bi-silicate, heated to 1,250°, was ground fine, and exposed first for 1.5 hr. to dry and then, for the same time, to moist carbon dioxide; in neither case was there any increase of weight, *i.e.*, no carbon dioxide had been absorbed. Treating caustic lime in the same way, after first heating it to 1,250°, showed that 0.2442 g. absorbed in 65 min. 0.0027 g. dry, and in 25 min. 0.0043 g. moist, carbon dioxide, or altogether 0.0070 g. (any moisture having been again expelled by heating to 110° C.), instead of the theoretical 0.19127 g. necessary to form calcium carbonate.

Attempts were made to test the natural calcium bi-silicate, wollastonite, but all the specimens available contained calcium carbonate.

While the differences in the behavior of the silicates and caustic lime when treated with dry and moist carbon dioxide are not very decided, they are sufficiently so to prove that in the silicates formed there are present only very small amounts of uncombined lime.

Having settled that calcium sulphate begins to be decomposed by silica at 1,000°, and is converted into silicate at a higher temperature, it was of interest to test a series of samples made up in molecular proportions corresponding to sub-, singulo-, bi-, and tri-silicates, by heating them in 10-min. periods for certain temperature-intervals, and thus obtain results which would be comparable among themselves and with the behavior of calcium sulphate alone or calcium sulphate and ferric oxide treated in a similar manner. The results are brought together in Table III., and represented graphically in Fig. 3.

They show that the rate of decomposition of calcium sulphate is proportional to the amount of silica used and to the temperature employed. Thus, at 1,200°, the amount of sulphur trioxide set free with the sub-silicate is 7.77, the singulo-silicate 12.78, the bi-silicate 17.50, and the tri-silicate 24.66 per cent.

This behavior may be explained by the more intimate contact of the particles of silica and calcium sulphate in the presence of much than of little silica. With liquid solutions, chemical reactions can easily be perfect, because the mixture of the reagents is complete, owing to the mobility of the mole-

TABLE III.—*Decomposition by Heat and Fluxes of Anhydrous Calcium Sulphate.*

Sample No.	Substance. Composition.	Weight.	Sulphur Trioxide.	Silica.	Temperature.	Time.	Loss. In Weight.	Loss. In Sulphur Trioxide.
		Grams.	Grams.	Per Cent.	Degrees C.	Min.	Grams.	Per Ct.
5	$CaSO_4$	0.1691	0.09941	.....	1,200	10	0.00035	0.35
					1,250	10	0.0013	1 31
					1,300	10	0.00435	4.38
					1,360	10	0.0177	17.80
					1,400	10	0.0362	36.45
6	$4\,CaSO_4 : SiO_2$, Sub-silicate.	0.1829	0.0978	9.09	1,000	10	0.00045	0.46
					1,100	10	0.0018	1.84
					1,150	10	0.00335	3.43
					1,200	10	0.0076	7.77
					1,250	10	0.01185	12.12
7	$2\,CaSO_4 : SiO_2$, Singulo-silicate.	0.2033	0.09783	18.15	1,000	10	0.0007	0.71
					1,100	10	0.0032	3.27
					1,150	10	0.0060	6.13
					1,200	10	0.0125	12.78
					1,250	10	0.02215	22.64
8	$CaSO_4 : SiO_2$, Bi-silicate.	0.2415	0.09838	30.72	1,000	10	0.0004	0.407
					1,100	10	0.00345	3.51
					1,150	10	0.0068	6.92
					1,200	10	0.0172	17.50
					1,250	10	0.0344	34.99
9	$2\,CaSO_4 : 3\,SiO_2$, Tri-silicate.	0.2430	0.08579	39.95	1,000	10	0.00065	0.76
					1,100	10	0.00635	7.4
					1,200	10	0.02115	24.66
					1,250	10	0.0384	44.76
10	$CaSO_4 : Fe_2O_3$, Calcium Ferrite.	0.4915	0.13296	.....	1,000	10	.....	.....
					1,100	10	0.00075	0.56
					1,150	10	0.0030	2.26
					1,200	10	0.011	8.27
					1,250	10	0.0417	31.37
					1,300	10	0.1271	95.60

cules and their partial dissociation, if the reagents are electrolytes. With solid substances which remain solid at the temperature of chemical reaction, this action can take place only at the points of contact. The more intimate this contact the more quickly will the reaction take place, and probably, also, the lower may be the temperature. With calcium sulphate and silica a contact of particles sufficiently close could be obtained by repeatedly triturating in an agate mortar for a considerable time, also by increasing the amount of one component—in the present experiments, of silica. Nevertheless, the light, soft, fluffy calcium sulphate did not permit a perfect mixing with the heavy, hard, and granular quartz particles.

The effect of a larger or smaller addition of silica is strikingly shown by the following experiment: A sample of 0.209 g. of a mixture containing 9.2 per cent. of SiO_2 and 0.11171 g. of SO_3 was heated for a considerable time at 1,250°, which caused the elimination of only 0.0335 g. of SO_3, or 34 per cent. of the SO_3-content, while the singulo-, bi-, and tri-silicate mixtures were completely decomposed at this temperature.

The decomposing effect of silica upon calcium sulphate may be said to be founded upon the stronger affinity at temperatures above 1,000° that silica has for lime than for sulphur trioxide, and upon its stability and infusibility, both permitting the expulsion and replacement of the more volatile and less refractory sulphur trioxide. The familiar decomposition of chlorides by phosphoric acid at elevated temperatures, with the formation of phosphates, may further illustrate the relation existing between silica and sulphur trioxide; the phosphoric acid is less volatile than the hydrochloric acid, and expels and replaces hydrochloric acid, which is the stronger of the two.

The results of the experiments with calcium sulphate and silica justify the deduction that the decomposition is due to the formation of silicate, and that the reaction is complete at a temperature varying from 1,250° to 1,300°, which is below the melting-point of the silicate. The concordant results of Day and Shepherd[21] and of Stein[22] show that the melting-point of calcium bi-silicate, $CaSiO_3$, lies at 1,512° and the formation-temperature at 1,250°, or 262° lower than the melting-temperature. Similarly, Day and Shepherd found the melting-temperature of calcium singulo-silicate, Ca_2SiO_4, to lie at 2,080°, while they were enabled to form it in a platinum crucible, as the formation-temperature lay below the melting-point of platinum, 1,753° (Waidner–Burgess). These authors make the general statement that under normal conditions the formation-temperatures of silicate-minerals lie below the melting-temperatures.

It is interesting to note that the decomposition of calcium sulphate by means of silica is practically finished at a temperature when the dissociation of calcium sulphate by heat alone is about to begin.

[21] *Journal of the American Chemical Society*, vol. xxviii., No. 9, p. 1089 (Sept., 1906).

[22] *Zeitschrift für anorganische Chemie*, vol. lv., No. 2, p. 159 (Aug. 27, 1907).

VI. Behavior of Anhydrous Calcium Sulphate with Ferric Oxide at Elevated Temperatures.

The information available as regards the behavior of ferric oxide at elevated temperatures is not quite satisfactory. Moissan[23] says that heating converts ferric into magnetic oxide, and that this fuses at the temperature of the electric arc. Recently Wüst,[24] in a paper upon the Theory of the Commutation Process, has given some data regarding the dissociation of ferric oxide. He found on heating ferric oxide in a vacuum that dissociation began at 600° C. According to his experiments, the dissociation-pressure of oxygen in ferric oxide at 930° is about 30 mm. Hg, or approximately 0.04 atmosphere. Supposing it to be the same at atmospheric pressure, ferric oxide cannot be dissociated at 930°, as the partial pressure of the oxygen of the air is approximately 0.21 atmosphere, and a metallic oxide cannot, through heat, give up any of its oxygen as long as the dissociation-pressure of the latter is smaller than the partial pressure of atmospheric oxygen.

If the dissociation-pressure of oxygen in ferric oxide at 1,500°, or 1,773° absolute, is calculated according to Nernst's formula,[25]

$$\log. p = -\frac{Q}{4.57\,T} + 1.75 \log. T + 2.8$$

in which p is the dissociation-pressure of oxygen and Q the calorific value, assumed as 195,600 g-cal.,[26] we obtain

$$\log. p = -\frac{195{,}600}{4.57 + 1773} + 1.75 \log. 1773 + 2.8 = \bar{2}.82102$$

$$\text{and } p = 0.06623 \text{ atm.}$$

This shows that in metallurgical processes carried on in carbon-heated furnaces, having a maximum temperature of 1,500°, no dissociation of pure ferric oxide can take place at atmospheric pressure.

In order to obtain experimental data for atmospheric pres-

[23] *Op. cit.*, vol. iv, p. 346.

[24] *Metallurgie*, vol. v., No. 1, p. 11 (Jan. 8, 1908).

[25] W. Nernst, *Theoretische Chemie*, Enke, Stuttgart, p. 704 (1907).

[26] Jüptner, *Journal of the Iron and Steel Institute*, vol. lxxv., p. 59 (No. III., 1907).

sure, pure ferric oxide was heated in purified dry air in the electric furnace. One sample of 0.2154 g. of Fe_2O_3 was heated in periods of 15 min., while the temperature was raised from 1,000° to 1,500°. The loss in weight at 1,500° was 0.0005 g., or 0.23 per cent.; repeated heatings at 1,500° showed no further change in weight; the charge was fritted. In a second sample of 0.200 g. of F_2O_3 and 0.05 g. of MgO, there was no loss in weight up to 1,490°. The conclusion is therefore that pure ferric oxide remains chemically unchanged when heated to 1,500° in pure dry air at atmospheric pressure, but it sinters.

In testing the effect of ferric oxide upon calcium sulphate a mixture corresponding to $CaSO_4$: Fe_2O_3 was heated in the platinum boat in pure dry air and the temperature gradually raised. Up to 1,100° C. there was no change in weight; at 1,100° gases began to be given off, which consisted of sulphur dioxide and oxygen; the evolution of gas increased with the rise of temperature, and appeared to be strongest at from 1,200° to 1,250°. At 1,200° the charge was fritted, at 1,225° it began to fuse at the edges, and at 1,250° it had melted to a ruby-colored liquid. This solidified upon cooling to a crystalline mass of calcium ferrite with radiating and cruciform black crystals having a brilliant adamantine luster. The photomicrographs represented in Figs. 4 and 5, p. 634, give two characteristic forms.

One charge of 0.3687 g., containing 0.1671 g. of $CaSO_4$, or 0.09974 g. of SO_3, lost upon heating for 10 min. at 1,250°, which resulted in complete fusion, 0.0997 g. of SO_3, or 100 per cent., *i.e.*, the calcium sulphate was completely decomposed. Another charge of 0.2363 g., containing 0.1088 g. of $CaSO_4$, or 0.06393 g. of SO_3, was charged into the furnace previously brought to 1,200°. After melting the charge at 1,250°, the loss was 0.0407 g. of SO_3, or 63.6 per cent.; prolonging the heating at 1,250° for 15 min. more gave a loss of 0.0638 g.; again the decomposition was complete. The second test shows that the charge will fuse without complete decomposition, a mixture of Fe_2O_3, CaO, and $CaSO_4$ being formed, and that the formation of ferrite continues in the fused charge until decomposition is perfect.

This calcium ferrite, which is little attacked by acids, forms

an interesting example of the so-called oxide slags. Calcium ferrites are mentioned by Stead and Risdale,[27] but no details are given as to their properties.

In order to determine the melting-point of calcium ferrite more accurately than was thought possible in using calcium sulphate and ferric oxide, a charge was prepared from lime and ferric oxide, the lime being obtained by heating Kahlbaum's calcium carbonate at 1,000° C. to constant weight. The mixture began to sinter at 1,200°, to fuse at 1,230°, became completely liquefied at 1,250°, and suffered no loss in weight. The solidified mass showed the same characteristics as that obtained from calcium sulphate and ferric oxide.

The following tests were carried through with a mixture of $CaSO_4$: Fe_2O_3 to obtain data for progressive rises of temperature with 10-min. time-periods. The results are given in Table III., and plotted in Curve VI. of Fig. 3, p. 639.

VII. Behavior of Anhydrous Calcium Sulphate with Lead Oxide at Elevated Temperatures.

The behavior of lead oxide at elevated temperatures has been recently studied by Doeltz and Graumann[28] and by Mostowitsch.[29] They found that lead oxide begins to be volatilized at 800° C., *i.e.*, 83° below its melting-point, and that the volatilization amounts to about 12.7 per cent. at 1,000°.

Lead sulphide does not act upon calcium sulphate, as has been shown by Doeltz.[30] The possible effect of lead oxide upon calcium sulphate has been the source of some speculation.[31] The formation of plumbates, as first assumed by Borchers,[32] is not maintained any more, as they are known to be unstable at roasting-temperatures. Thus, Le Chatelier[33] says that the dissociation-pressure of 2 CaO, PbO_2 at 880° C. is 47 mm. of Hg, and at 1,100°, 940 millimeters.

A mixture of the proportions PbO: $CaSO_4$ was heated in dry purified air. Up to 700° C. no change in weight took

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[27] *Journal of the Iron and Steel Institute*, vol. xxx., pp. 222, 235 (No. I., 1887).
[28] *Metallurgie*, vol. iii., No. 12, p. 406 (June 22, 1906).
[29] *Op. cit.*, vol. iv., No. 19, p. 647 (Oct. 8, 1907).
[30] *Op. cit.*, vol. ii., No. 19, p. 460 (Oct. 8, 1905).
[31] Tandler, *Chemiker Zeitung*, vol. xxxii., No. 35, p. 437 (Apr. 29, 1908).
[32] *Metallurgie*, vol. ii., No. 1, p. 1 (Jan. 8, 1905).
[33] *Comptes rendus*, vol. cxvii., No. 2, p. 109 (July 10, 1893).

place; at 700° lead oxide began to be volatilized instead of at 800°, as found by Doeltz and Graumann. With rising temperature the quartz heating-tube became filled with vapors of lead oxide, but the absorption-tube showed neither sulphur dioxide nor trioxide. The loss in weight increased rapidly with the heat. At 1,200° the experiment was stopped, as the vapor of lead oxide, combining with the quartz tube to a viscous silicate, caused the platinum boat to adhere to the tube. A sample of 0.6109 g. of mixture, containing 0.3792 g. of PbO, heated for 30 min. at 1,200° C., lost 0.3750 g. in weight, which is equal to 94.15 per cent. of the amount of lead oxide

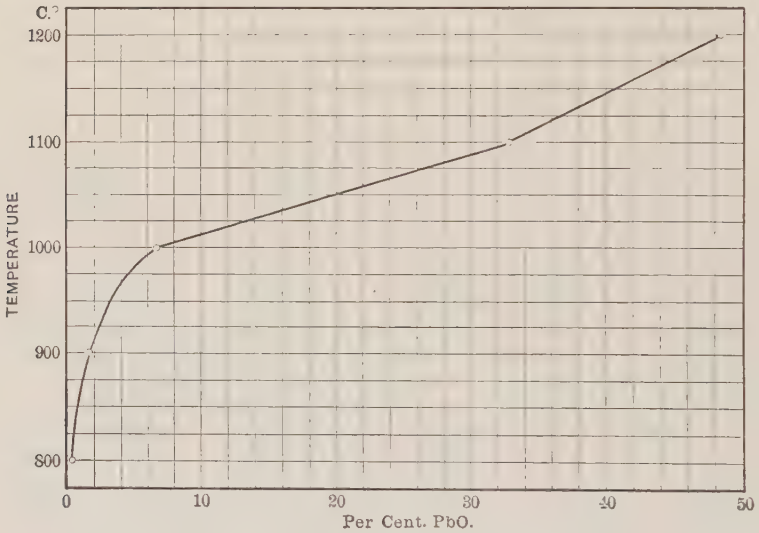


FIG. 6.—VOLATILIZATION OF LEAD OXIDE.

present. A qualitative examination of the residue showed the presence of undecomposed calcium sulphate and of some lead oxide; it gave no alkaline reaction with phenolphtalein, *i.e.*, it contained no caustic lime; the solutions in the absorption-bottles remained clear; hence, neither sulphur dioxide or trioxide was given off, and no calcium sulphate was decomposed.

Details of the tests carried on for 15-min. periods within a range of temperature of 800° to 1,200° are given in Table IV. and plotted in Fig. 6. The course of the curve between 1,100° and 1,200° is not absolutely correct, as some of the viscous lead silicate which had formed adhered as a very thin

transparent glaze to the platinum boat. The only means of detecting it was a slight roughness of the platinum surface, and no means except rubbing was available for removing it.

TABLE IV.—*Behavior of Anhydrous Calcium Sulphate with Lead Oxide.*

Sample No.	Character of Substance.	Weight of Substance.	Weight of Lead Oxide in Substance.	Time.	Temperature.	Loss in Weight.	Loss in Lead Oxide.
		Grams.	Grams.	Min.	Degrees C.	Grams.	Per Cent.
11	$CaSO_4$;PbO	0.6109	0.3792	15	800	0.00065	0.17
					900	0.0056	1.48
					1,000	0.02485	6.55
					1,100	0.1246	32.86
					1,200	0.179	47.20

VIII. SUMMARY OF RESULTS.

1. Hydrous calcium sulphate loses its combined water at about 900° C.

2. Anhydrous calcium sulphate heated at atmospheric pressure in pure dry air remains unchanged up to 1,200° C. At 1,200° it begins to be dissociated, with the separation of sulphur dioxide and oxygen, as at this temperature sulphur trioxide cannot exist as such.[34] The rate and degree of dissociation vary with the time and the temperature.

Calcium sulphate fuses at about 1,360° C. and is more or less decomposed; the resulting product is a mixture of calcium oxide and sulphate; the proportions of the components vary with the time and temperature of the preliminary heating.

3. Silica decomposes calcium sulphate at an elevated temperature; the decomposition begins at 1,000° C. and is finished at 1,250°; the product contains only calcium oxide and silica; the reactions taking place may be expressed by

$$CaSO_4 + SiO_2 = CaSiO_3 + SO_2 + O$$
$$2\ CaSO_4 + SiO_2 = Ca_2SiO_3 + 2\ SO_2 + O, \text{ etc.}$$

The examination of the product leads to the conclusion that the formation-temperature of calcium silicate lies below its melting-temperature.

4. Pure ferric oxide heated at atmospheric pressure in pure dry air is not dissociated at 1,500° C.

[34] Knietsch, *Berichte der deutschen chemischen Gesellschaft*, vol. xxxiv., No. 17, p. 4069 (1901); *Mineral Industry*, vol. x., p. 605 (1901).

5. Pure ferric oxide begins to decompose calcium sulphate at 1,100° C.; the decomposition is finished at 1,250°; ferric oxide requires on the whole higher temperatures to decompose calcium sulphate than does silica, but the decomposition proceeds at a more rapid rate. The process taking place may be expressed by the reaction

$$CaSO_4 + F_2O_3 = CaO.\ F_2O_3 + SO_2 + O$$

The calcium ferrite formed fuses at 1,250° C.; it has, however, the property of dissolving calcium sulphate and of decomposing it, which indicates the existence of ferrites other than given by the formula.

6. Pure lead oxide has no decomposing effect upon calcium sulphate.

The leading data regarding the decomposition of anhydrous calcium sulphate by heat and by the fluxes silica and ferric oxide from Table III. are assembled in Table V.

TABLE V.—*Decomposition of Anhydrous Calcium Sulphate by Means of Heat and Fluxes.*

Time of Heating.	Temperature.	Loss of SO_3, in Per Cent. of SO_3 Contained in Charge.					
		$CaSO_4$.	4 $CaSO_4$: SiO_2 Ca_4SiO_6.	2 $CaSO_4$: SiO_2 Ca_2SiO_4.	$CaSO_4$: SiO_2 $CaSiO_3$.	2 $CaSO_4$: $3SiO_2$ $Ca_2Si_3O_8$.	$CaSO_4$: Fe_2O_3 $CaFe_2O_4$.
Min.							
10	1,000	.....	0.46	0.71	0.47	0.76	.....
10	1,100	.....	1.84	3.27	3.51	7.40	0.56
10	1,150	.....	3.43	6.13	6.92	.....	2.26
10	1,200	0.35	7.77	12.78	17.50	24.66	8.27
10	1,250	1.81	12.12	22.64	34.99	44.76	31.37
10	1,300	4.38	.....	.....	.....	.....	95.36
10	1,360	17.80	.....	.....	.....	.....	.....
10	1,400	36.45	.....	.....	.....	.....	.....

IX. APPLICATION OF THE EXPERIMENTAL RESULTS TO SOME METALLURGICAL PROCESSES.

1. *The Carmichael-Bradford Process.*—The gypsum added to the charge of raw sulphide lead-ore, characteristic for this mode of blast-roasting, is decomposed mainly by silica and ferric oxide; the sulphur dioxide resulting from the dissociation of sulphur trioxide set free enriches the gases.

Gypsum has no decomposing effect upon galena.

The lead sulphide of the charges is oxidized by the oxygen of the air blown into it; the sulphur trioxide dissociated into

sulphur dioxide and oxygen at the temperature of decomposition of calcium sulphate cannot have any oxidizing effect in the hot parts of the charge; in the upper cooler region of the converting-pot there is a possibility of some sulphur trioxide being formed from sulphur dioxide and oxygen with the assistance of a catalyzing effect of the charge, but even here the newly-formed sulphur trioxide can hardly have any oxidizing effect, as the equations

$$SO_3 = SO_2 + O - 22{,}600 \text{ cal.}$$
$$PbS + 3\ O\ (air) = PbO + SO_2 + 104{,}000 \text{ cal.}$$

point toward the strongly exothermic direct oxidation by means of the excess of free oxygen present in the gases.

The decomposition of calcium sulphate with the simultaneous dissociation of sulphur trioxide,

$$CaSO_4 = CaO + SO_3 - 95{,}600 \text{ calories},$$
$$SO_3 = SO_2 + O - 22{,}600 \text{ calories},$$
$$- 118{,}200 \text{ calories},$$

represents two strongly endothermic reactions which must be favorable to the roasting of galena, as they counteract any tendency to local overheating and consequent fusion of the charge.

Further, the sulphur dioxide set free has a loosening effect upon the charge and thus acts favorably upon the process. From the thermal considerations it appears that in blast-roasting galena, calcium sulphate has a more favorable effect upon the result than calcium carbonate, which, according to the reaction, $CaCO_3 = CaO + CO_2 - 45{,}150$ cal., absorbs less heat.

2. *Ordinary Oxidizing-Roast.*—In the ordinary oxidizing-roasting of ores the temperature is usually not raised to above 900° or 1,000° C. As gypsum is not decomposed at this temperature, its presence can but have the effect of increasing the percentage of sulphur in the roasted charge.

3. *Oxidizing-Roast of Calcareous Blende.*—In the oxidizing-roasting of calcareous blende the calcium carbonate is converted into calcium sulphate; and the latter, not being decomposed at the temperature of from 1,200° to 1,250° C. attained in the operation, might be expected to increase the sulphur-content beyond the degree it actually does were it not for the presence of small amounts of silica in the gangue and of isomorphous

iron sulphide in the blende converted into ferric oxide, both of which favor the decomposition of calcium sulphate, and are thereby helpful in the diminution of the sulphur-content.

4. *Reduction of Roasted Calcareous Blende.*—On the other hand, the presence of large amounts of isomorphous iron sulphide in blende, as in many marmatites, has a harmful effect upon the reduction of roasted calcareous blende on account of the low temperature (1,250° C.) at which calcium ferrite fuses. The reduction of zinc oxide in practice is carried on at a temperature of 1,300° C.; the fused calcium ferrite, which is very fluid, will penetrate the charge, hinder reduction, and make rich residues; further, it will have a corrosive effect upon the retort-material.

5. *Knudsen Process for Pyritic Smelting.*[35]—In this process, it will be remembered, copper-bearing pyrite is smelted in a converter lined with dolomite. Practical experience has shown that the more basic—*i.e.*, the richer in iron—the slag, the more quickly will the lining be slagged, while slags with a reasonably high percentage of silica do not corrode the lining as rapidly. The reason for this peculiar fact now finds its explanation in the formation of readily-fusible calcium ferrite. The experiment given on p. 646 has shown that magnesium ferrite does not form at a temperature as high as 1,500° C.; hence, the richer in magnesia the dolomite, the more resistant will it prove to slagging.

6. *Basic Steel Processes.*—The facts stated in the preceding paragraph regarding the ease with which calcium oxide is slagged by ferric oxide, and the difficulty of forming a magnesium ferrite, are important as regards the lining of basic-steel furnaces. Here it has also been found in practice that magnesia resists the corrosive effect of ferruginous slag much better than does lime.

7. *Rôle of Ferruginous Slags in Non-Ferrous Smelting.*—The fluxing effect that basic slag rich in iron from copper- or nickel-refining furnaces has upon the blast-furnace charge of acid ores, may be explained in part by the fusibility and fluidity of calcium ferrite. This oxide slag readily dissolves

[35] Schiffner, *Berichte V. internationaler Kongres angewandte Chemie*, vol. ii., p. 112 (1904); Knudsen, *Engineering and Mining Journal*, vol. lxxvii., No. 19, p. 757 (May 12, 1904).

the less-fusible component parts of the charge, and helps to form at the desired temperature a slag of the required fluidity. The general principle of slag-formation by solution has been proved experimentally in a different way by Doelter,[36] who found, in observing slag-formation under the microscope, that the process was one of solution, in which the mineral having the lower melting-point, after fusing, dissolved the one having the higher.

In conclusion, we desire to express our obligation to Prof. C. L. Norton, in charge of heat-measurement at the Massachusetts Institute of Technology, in whose laboratory the research was carried on.

Philippine Coal-Fields.

BY J. B. DILWORTH, PHILADELPHIA, PA.

(Chattanooga Meeting, October, 1908.)

Outcrops of coal have been discovered in many localities in the Philippine archipelago, and practically all of the larger islands contain deposits of this mineral. Very little prospecting has been done to determine the value of the various coal-bearing areas, but so far as known, the largest and most promising fields are found: (1) in northern Cebu; (2) in the small island of Batan, a part of the province of Albay in southeastern Luzon; (3) in the somewhat larger island of Polillo, off the east coast of Luzon, and (4) in southern Mindoro, near the small coast town of Bulalacao.

Northern Panay, south-central Cebu, and various localities in southern Luzon, Masbate, Negros, and Mindanao also afford evidences of the existence of coal-beds, the extent and value of which have not yet been proved. The general position of these islands and districts is shown on the map of a portion of the Philippine Islands, Fig. 1.

Some of these coal-fields have been known for many years (coal was discovered in Cebu in 1827 and in Batan in 1842),[1]

[36] *Physikalisch-chemische Mineralogie*, Barth, Leipsic, p. 127 (1905).

[1] *The Coal-Measures of the Philippines*, by Charles H. Burritt, Government Printing Office, Washington, D. C. (1901).

and several futile attempts were made by the Spaniards to operate mines in these districts. But owing to many circumstances (chief among which were indifferent management, the political unrest of the islands, natural difficulties of transportation, and those inherent in the mining of steeply-inclined and somewhat irregular seams), no considerable output was ever obtained from any of these mines, and the fuel required in the many departments of industrial and political life has continued to be drawn largely from Australia and Japan.

These conditions exist to a great degree at present, and though much interest in the coal-deposits of the Philippines has been aroused within the last few years, and considerable development-work has been carried on, there is now but one mine regularly supplying coal to the trade. This mine, operated by A. U. Betts, on the SE. coast of Batan Island, began in a very primitive way in the latter part of 1906, and when visited in the spring of 1907 had a daily capacity of about 25 tons. All of this output was taken by local steamers and had no appreciable effect on the general market.

Markets.—The present annual consumption of coal in the Philippines approximates 350,000 long tons. Of this amount, 300,000 tons is imported from Australia and Japan for use by inter-island steamers, railroads, factories, ice- and distilling-plants, etc.; the remaining 50,000 tons is high-grade steam-coal, chiefly from the Pocahontas field in Virginia and West Virginia, which is brought "around the Horn" for use by the United States Navy, and with which the native coal cannot hope to compete.

Consumption is increasing yearly as new railroads are completed, new industrial plants installed, and both foreign and inter-island commerce increased under the aggressive policy of American interests now developing in all parts of the archipelago.

It is expected that, within the next four years, 650 miles of railroad will be operating in Luzon and the Visayan, or central, belt of islands, a mileage more than four times as great as that now in use. Several hundred steamers call at the various ports each year, most of which now coal in foreign harbors with high-priced fuel of a grade not greatly superior to that found in several of the Philippine fields.

FIG. 1.—MAP OF A PORTION OF THE PHILIPPINE ISLANDS, SHOWING ISLANDS AND COAL-FIELDS EXAMINED.

An annual consumption of more than 500,000 tons within the next five years seems, therefore, to be a conservative estimate, which will assure a home-market for the output of several large coal-mines.

The native coal is somewhat inferior to imported coals, owing to its generally higher percentage of moisture and consequent lower calorific value; nevertheless, it is believed that the better grades can be substituted in every place where Australian and Japanese coal is now used.

This fact, in connection with the well-based estimate that native coal can be delivered in the principal ports of the archipelago at a price of from 50 to 60 per cent. of the lowest prices of competing foreign coals, leads to the conclusion that well-managed mining operations in the better native fields should have no difficulty in disposing of the output, and eventually in dominating the home-market, in addition to the possibilities of building up a subordinate trade with the neighboring ports of Singapore, Hongkong, and Shanghai, all consumers of large quantities of Australian and Japanese coal.

Native Labor.—There has been considerable speculation on the part of those interested in mining and other industrial operations in the Philippines as to the suitability of the Filipino as a laborer. Until recently the opinion prevailed that he was too indolent and irresponsible and had too many "feast-days" on his calendar to make an efficient and reliable miner or day-laborer.

This early belief is fading out in the light of experience, and large and important contracts for harbor-improvements, railroad-construction, and other public works are being successfully carried out by those employing thousands of natives as unskilled laborers. In the old Spanish mines natives were employed extensively, and in the preliminary prospecting and mining now being conducted in various places Filipinos are employed both underground and on the surface.

Undoubtedly, the native miner will not prove as strong or as efficient as the Japanese or American miner; but neither will he expect as much pay, and, if fairly treated and directed by white bosses, he will probably put out more coal per dollar of wages received than the more skillful miner, and be a peaceful and persevering worker. Instances can be cited of individual natives who proved to be exceptionally intelligent, and were

able creditably to fill responsible positions requiring skill and judgment.

Wages vary considerably in different localities, due to the varying density of population and the natural resources of the district. In the island of Cebu, which is very thickly settled by relatively poor inhabitants, the unskilled laborer receives 30 cents per 10-hr. day, while the more responsible positions—such as watchmen, firemen, blacksmith and carpenter helpers—pay from 50 cents to $1 per 10-hr. day. In southern Luzon, where there is a great demand for laborers in the hemp-plantations and warehouses, wages are about double those paid in Cebu.

Mining-Costs.—It is impossible to estimate very accurately what will be the ultimate cost of mining in any one field, both by reason of the insufficient data as to the structure and character of the rock-strata and coal-seams, and because no accurate costs of the various items involved can be obtained from mines in actual operation under like conditions. Assuming a steeply-dipping seam of 7 ft. thickness, modern mining-methods, wages approximating those now paid in Cebu, and a daily output of 500 tons, it is believed that coal can be delivered at the mine-mouth for $1 per ton.

Transportation- and storage-charges would vary with each locality, and would have to be computed especially for the case in hand. As a general rule, however, coal from any of the known fields in the islands should be put in Manila harbor at a cost not exceeding $2.50 a ton, where it would have to compete with foreign coal, of 25 per cent. greater calorific power, selling under normal conditions at from $4 to $4.50 per ton. (Since the Russo-Japanese war, Australian coal has monopolized the Philippine market, with a consequent increase in price to $5.90 per ton in 1906.) On this basis of relative cost and fuel-efficiency, it seems as though successful competition with foreign fuel would not be difficult.

Lantauan Coal-Field, Cebu Island.

Cebu is a long, narrow island, situated near the geographical center of the Philippine archipelago. The central portion is occupied by a rugged mountain range running the entire length of the island, and varying in altitude from 2,000 to 3,000 ft. The Lantauan coal-field is situated in the eastern spurs and

foot-hills of this range, on the head-waters of the Danao river, a small stream which enters the sea at the town of Danao, about 50 miles from the northern end of the island.

The province of Cebu, which embraces the entire island, is one of the most densely populated in the Philippines, its total number of inhabitants, in 1903, being reported as about 650,000. Cebu City, the capital, has a population of 35,000, and is second only to Manila in size. Many of the natives live in the towns and cities which line the coast, while great numbers are the ignorant, simple-minded country folk who live in frail palm-huts, built on the borders of corn- and banana-patches which dot the mountain-sides.

In the past the principal means of inter-communication and transportation was sail-boats on the sea, and carts or sleds on land, principally the former, as there are but few roads along the coast and none across the central mountain range, so that the towns on the east coast are practically farther from those on the west than from the neighboring islands of Bohol, Leyte, and Samar. The new railroad of the Philippine Railway Co., which parallels the eastern coast-line between the towns of Danao and Argao, is being rapidly pushed to completion.

Coal has long been known in Cebu, and coal-seams have been found in several widely-separated localities. The most promising discoveries have been made in the Danao River area; and the old Danao-Compostella operations—perhaps the most highly-developed of the Spanish coal-mining enterprises in the Philippines—were located just south of the Lantauan field, and probably in the same series of coal measures.

The Lantauan coal-field proper comprises two rather wide, flat valleys, with an average elevation of about 1,100 ft. These valleys—the Camumayan in the northwest and the Cabezolino in the southeast—are separated, topographically and geologically, by the abrupt and narrow Lantauan ridge, rising to elevations of from 1,400 to 1,900 ft. above sea-level.

The dividing ridge is composed of basic, igneous, diorite rock, with a capping on its higher knobs of coralline limestone, sometimes as much as 500 ft. thick. The rocks of the valleys are sedimentaries, comprising conglomerates, sandstones, and shales, with inter-stratified coal-seams, which have been generally referred to the Eocene epoch of the Tertiary period.

In the Camumayan valley the structure is that of a wide basin, or trough, with its long axis parallel to the Lantauan ridge. The NW. limb is rather more regular than the opposing one, and the coal-seams contained therein are more uniform in character and thickness.

The several coal-beds of this valley, averaging about 9 ft. thick, are usually free from partings, and have a general NE–SW. strike and a 60° SE. dip.

In the Cabezolino valley the coal-seams are neither so numerous nor so thick as in the adjoining Camumayan district. They average about 8 ft. thick, and strike slightly east of north, with an eastward dip of about 35°.

All of the coal-seams in the Lantauan coal-field yield coals of very similar physical and chemical properties. These coals are usually hard and strong; density about 1.3; of jet-black color, with rather dull luster; possess a marked "block" cleavage, though the larger blocks usually break with a conchoidal fracture into sharp-edged pieces; show practically no trace of vegetable structure; are readily ignited, and burn without evidence of coking, but with a clear, abundant flame, yielding a fine light ash.

The chemical quality of the coal from the principal beds of the two valleys is shown by the following average analysis, computed from analyses of 16 samples taken personally, made in the laboratory of the Bureau of Science at Manila:

Lantauan Coal-Field, Average Coal-Analysis.

	Per Cent.
Water,	17.20
Volatile matter,	34.69
Fixed carbon,	44.60
Ash,	3.51
Sulphur,	0.60
B.t.u.,	9,654

This analysis shows that the coals in this field belong to a class, intermediate between the true bituminous coals and the true lignites, for which the name "sub-bituminous" has been adopted by the U. S. Geological Survey. While the moisture is quite high, the coal yields notably low percentages of ash and sulphur, and has a calorific value not greatly below that

of the Australian and Japanese coals now on the Philippine market.

Australian and Japanese Coals, Average Analysis.

Australian Coal: 12 Samples.	Per Cent.	*Japanese Coal: 25 Samples.*	Per Cent.
Water,	2.59	Water,	2.26
Volatile matter,	34.38	Volatile matter,	38.43
Fixed carbon,	53.23	Fixed carbon,	50.23
Ash,	9.80	Ash,	9.08
Sulphur,	0.60	Sulphur,	0.78
B.t.u.,	12,512	B.t.u.,	11,992

These foreign coals are distinctly superior to that from Cebu; but the gain in having a relatively low moisture-content of the foreign fuels is partly offset in their higher percentages of ash. In calorific power they exceed the native coal by about 25 per cent.

Batan Island.

This island, with an area of only 40 sq. miles, is much smaller than Cebu. It is the central one of three islands forming the northern boundary of Albay gulf, which indents the extreme southeastern end of Luzon, the great northern island of the Philippines.

Politically, Batan is a part of the province of Albay, and its scant population, of about 2,000 souls, belongs to the Bicol tribe, which is found in large numbers on the mainland.

There is no large town on the island; most of the inhabitants are gathered in a few small fishing-villages along the coast; but Legaspi, 16 miles SW. of Batan Island on the western end of Albay gulf, is a prosperous and important port.

Albay is one of the greatest hemp-growing provinces in the islands, and the extensive demand for laborers on the plantations, presses, and docks has so benefited the natives that labor can only be procured at a price considerably above that paid in the less-prosperous communities.

Batan is quite mountainous, like most of the islands in these seas, and the rim of ridges which follows the coast-line rises in several places to altitudes of from 800 to 1,400 ft. These ridges are generally clothed with virgin forests containing much valuable timber. The streams are small and generally intermittent, as might be expected on so limited an area, and

most of the fresh water used by the inhabitants is drawn from wells and springs.

Coal has been known to exist in Batan since the first half of the last century; indeed, as early as 1842 the Spanish Governor Velarde, of Albay, operated a mine in its NE. part. More recently, in the five years preceding the American occupation, other Spanish mining-interests took up claims in various parts of the island, and did some little development-work on them.

In 1902 the United States government set aside the entire western end of Batan as a military reservation, and in the two succeeding years a large amount of prospecting by diamond-drills and short drifts was done in the SW. part of this reservation under the supervision of H. L. Wigmore, First Lieut., Corps of Engineers, U. S. A.

Almost the whole of Batan Island is composed of sedimentary, stratified rocks, which have been classified by the palæontologists of the Mining Bureau, Manila, as of Tertiary age.[2] Associated with these strata are intrusive igneous rocks, exposed in a few places along the coast, and thick beds of coral-line limestone, usually capping the higher ridges and spurs.

In the western half of the island the strata have a general northward dip, varying from almost 0° to 35,° or even more. Several coal-beds of commercial thickness have been opened by drill and prospect-pit; but the general impression produced by the examination of records of past developments and of present exposures is that the region is disturbed and, in places, faulted; also that the coal-seams are variable and lacking in continuity over large areas.

The quality of the coal is indicated by the following average of 20 analyses of drill-cores and outcrop-samples, made by the Bureau of Science, Manila:

Batan Island Coal, Average Analysis.

	Per Cent.
Water,	5.75
Volatile matter,	41.04
Fixed carbon,	47.64
Ash,	5.57
Sulphur,	0.72
B.t.u.,	11,549

[2] Warren D. Smith, The Coal Deposits of Batan Island, *Bulletin No. 5, Mining Bureau, Manila* (1905).

This is a very fair grade of bituminous coal, with relatively low moisture and ash, and a calorific power quite comparable with that of coals from Australia and Japan.

Considering the eastern half of Batan, the strata are again found to have a northward dip, which amounts to 30° ± on the north side, falls to almost 0° near the center, and increases again to about 10° along the southern coast. The thickness of these sedimentaries probably exceeds 2,000 feet.

In the northern part of this division occur several rather thin coal-seams, which are apparently regular in structure and bed-sections. It was in this series of beds that the Spanish mines were located. The failure of these operations seems to have been due more to the unstable condition of the government and to ignorant, injudicious, and unfaithful management than to any inherent faults in the coal-beds themselves.

Several thin seams have been exposed in the interior of the island, but nothing of importance is shown until the southern coast is reached. Near the village of Batan, a small operation, owned by A. U. Betts of Legaspi, is working a 2 ft. 11 in. seam of clean coal about 20 ft. above sea-level at the slope-mouth, and dipping northward 10°. When visited, early in 1907, the mine had been running but a few months, and was being operated entirely without power; its daily capacity was about 25 tons. Below the seam worked is another somewhat thicker seam which has been but sparingly opened. No evidences of folding or faulting were observed in this district, and it is probable that very considerable quantities of coal might be won from this and the upper bed.

An average analysis of 19 coal-samples from this eastern half of Batan Island is:

Batan Island Coal, Average Analysis.

	Per Cent.
Water,	16.44
Volatile matter,	41.62
Fixed carbon,	36.15
Ash,	5.79
Sulphur,	1.59
B.t.u.,	9,148

The coals of the eastern division of Batan Island are much higher in moisture than those from the western division, where

the dynamic action which disturbed and folded the beds, also displayed itself in the lowered water-content of the coals.

Burdeus Coal-Field, Polillo Island.

As shown in Fig. 1, Polillo is a small island about 27 miles long and 9 miles wide, lying in the Pacific ocean just off the east coast of Luzon in about the latitude of Manila. It is peculiarly isolated by being off the regular lines of inter-island steamers, and separated by a channel 15 miles wide from the mainland, which is itself but sparsely settled and quite effectually separated from the populous district of western Luzon by an almost trackless mountain barrier. The town of Infanta, just across the channel from Polillo, is the point from which travel usually proceeds to the island.

The inhabitants are practically all Tagalogs, the most advanced of the Philippine tribes. In 1898–1899, their number was estimated at 1,400, over half of whom probably lived in the "pueblo" of Polillo on the west coast, virtually the only town in the island.

A high central ridge runs the full length of the island, its projecting spurs often extending to the coast and forming the water-sheds of many small streams. The highlands are covered with forest, as in Batan, but here the trees usually afford only the lower grades of lumber.

The Burdeus coal-field lies in the east-central portion of the island on the drainage of Burdeus and Aloeon creeks. It is about 1 mile wide by 5 miles long—its longer extension being in a N–S. direction—and occupies the eastern foot-hills of the main ridge, separated from the sea by a coastal plain a mile wide.

The region is one of great geologic disturbance, having been subjected to an E–W. pressure which folded the measures into a series of anticlines and synclines, the action evidently having been accompanied in places by faulting. The general strike of the measures is N–S., and the dip is of every degree from 0° to 90°.

Numerous coal-outcrops have been uncovered at various points, but no extensive development or thorough exploitation of the beds has yet been undertaken. The seams vary in thickness from a few inches to 7 ft.; but the meager evidence now at hand indicates that the larger beds have no great persistency,

and are probably so irregular in bed-section and so folded as to be unattractive as mining propositions.

Numerous samples of Polillo coal have been analyzed by the Mining Bureau, Manila, and the following is an average of 10 of them, all of which probably came from the Burdeus field:

Polillo Coal, Average Analysis.

	Per Cent.
Water,	5.58
Volatile matter,	41.20
Fixed carbon,	49.15
Ash,	4.07
Sulphur,	0.29
B.t.u.,	11,738

All practical and laboratory-tests of this coal show it to rank in quality with the best in the Philippines. It is the equal chemically of that from western Batan, and is but little inferior to the Japanese fuel now on the market. But in this field, as in western Batan, geologic disturbances have rendered the coal-seams structurally difficult to mine economically on a large scale.

A Labor-Chart for the Management of Mining and Milling Operations.

BY JOSEPH MACDONALD, GUANAJUATO, MEXICO.

(Chattanooga Meeting, October, 1908.)

STRIPPED of its romantic possibilities, mining is a commercial business, carried on for the profit there is in it; and the business of the manager, in its ultimate analysis, is to make the profit as large as possible.

The general manager of a mining company should keep himself personally familiar with all the details of its operations. If he fails to do so, chaos results, labor becomes inefficient, costs go up, and profits go down. Nowhere is this emphasized more than in Mexico.

If the mining operations are extensive and complex, involving, besides the work of mining the ore, its transportation and treatment by various milling operations, and the marketing of the products, the business, to be carried on properly, must be

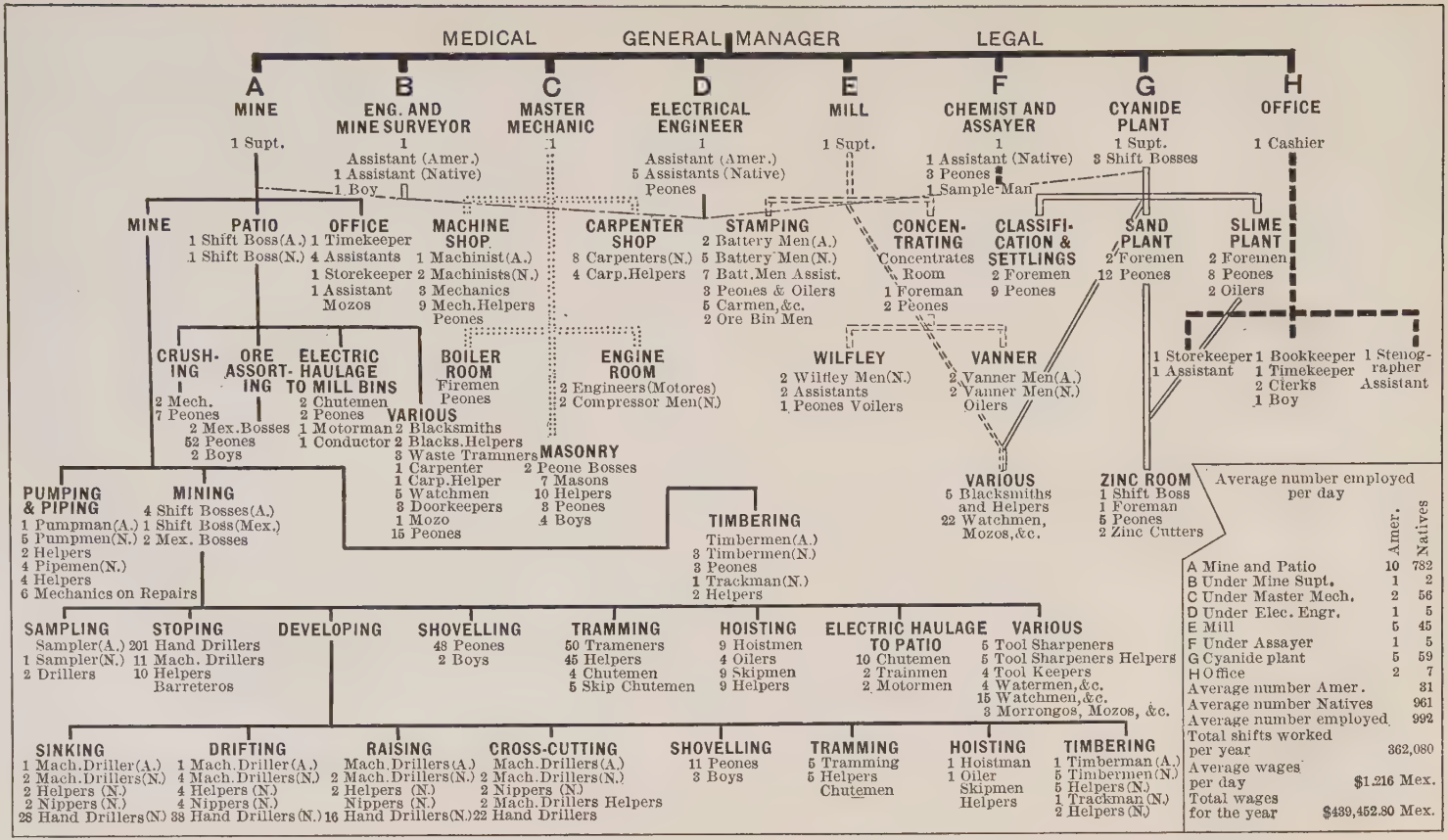


FIG. 1.—GUANAJUATO CONSOLIDATED MINING & MILLING CO. STAFF AND DISTRIBUTION OF LABOR FOR 1907.

divided into a number of main departments, which are sub-divided into minor departments. Each main department should be under the immediate charge of a superintendent, or a foreman, and each minor department under a shift-boss. Organized in this way, and with all the principal centers of activity connected by telephone, the manager can keep in touch at all times with the details of every branch of the business, and be in a position to perform effectively the full duty of manager.

The total cost of mining operations may be broadly divided under the heads of Labor, Supplies, and Marketing. Of these, it will be found, on averaging a number of cases, that the cost of labor will be about two-thirds of the total cost. The importance, therefore, of an efficient labor-department will be apparent. This department must be sub-divided and organized differently, according to the peculiar conditions at each mine, and no one system, however well suited for one particular concern, will exactly fit another. But any standard system may be modified to suit any particular case.

Each foreman of a given department should be furnished with suitable blank forms, on which he should record, at the end of every shift, the main details of the operation of his department. These blanks, when filled out, pass through the superintendent's hands, and, with his remarks and explanations, are presented to the manager for examination and filing. Thus, within 24 hours, the record of the work done in every departmental sub-division of mine, or mill, together with its cost, is before the manager for study and comparison with previous records.

It is not always possible, in fact it is very difficult, to get foremen, or shift-bosses, who are in every way competent to supervise the departments, but the best men may be selected for this purpose, who, if properly supervised and encouraged by the manager in the work, will give profitable service.

The work of the mine-manager will be facilitated by devising a labor-chart which will show diagrammatically the various departments of labor, the number and class of men employed, the cost, and the method of supervising each class.

The foreman of each department should have a labor-chart pertaining to his own special department, and all charts

of the various superintendents, combined on one sheet, should make up the manager's chart, which will serve also as a "flow-sheet" of authority in the organization. A labor-chart may show the distribution of labor for a month, or a year, or any other period.

The labor-chart I used for the Guanajuato Consolidated Mining & Milling Co. was included in my annual report to the directors covering the operations of that company during 1907, and with the permission of the President of the company, F. G. Corning, I submit this chart, Fig. 1, for the purpose of illustration. It shows the labor-classification and the cost for the operations of the company during the year. The chart, in its original form, was printed in colors, so that the eye could readily distinguish the ramifications of the various sub-divisions. The general division was in black and blue; mining, in red; milling, in purple; cyaniding, in light-brown; office work, in dark-brown. In Fig. 1, each color has been replaced by a distinctive line. The method of arranging the details of labor-classification might be more advantageously shown than is done in the chart, but it is hoped that it will serve as a suggestion for a system that renders more efficient the management of mining operations on a large scale.

Luther, Körner, Humboldt, and Swedenborg.

BY R. W. RAYMOND, NEW YORK, N. Y.

(New York Meeting, February, 1908.)

FOUR portraits have recently been hung in the rooms of the Institute, in recognition of four illustrious men with whom we, as mining engineers and metallurgists, may claim fellowship.

LUTHER.

Martin Luther is often spoken of as a miner's son; but it is less generally known that John Luther, his father, was, in a small way, a metallurgist as well as a miner, and that Martin Luther himself, in his youth, worked in a forge or bloomary established by his father at Mansfeld, Saxony. Since this is the only manual occupation on his part of which we have any record, it is fair to assume that the great scholar and reformer derived from his work in this department the practical familiarity with the conditions of human labor so abundantly evinced in the homely illustrations which accompany even his serious arguments, and in the witty apothegms of his famous "table-talk." At the same time, I must confess that I have not been able to find in any of the works of Luther distinct allusions to mining or metallurgy.

KÖRNER.

Theodor Körner, a student at the Mining Academy of Freiberg, Saxony, threw himself into the German War of Freedom in the first decade of the 19th Century. He was a poet of no mean rank; and his "Song of the Sword" is one of the great lyrics of literature. This poem was written but a few days before he died in battle. In many respects, Körner may be compared with our American hero Theodore Winthrop, who similarly met a glorious fate, and left behind him prophetic indications of genius, of which his premature death prevented the complete fulfillment. I am informed by a member of the Institute that Körner's admirable collection of minerals is still preserved at Dresden.

VON HUMBOLDT.

Friedrich Heinrich Alexander von Humboldt was born in 1769, and studied at two German universities. He published his first book, on the *Basaltic Rocks of the Rhine*, in 1790. In the following year he went to Freiberg, where those great men, Werner and Leopold von Buch, were his teachers. In 1793 he published his treatise on the *Subterranean Flora of Freiberg*, and from 1792 to 1797 he was employed as a mining official at Baireuth. This was the only practical drill which he ever experienced; and while it is not possible to ascribe supreme influence, in the career of such a universal genius, to any one episode, I think it fair to say that in many, if not all, departments of Humboldt's cyclopedic knowledge and world-wide exploration and study, the effect of his training as a geologist and mining engineer may be traced. To this cause may certainly be ascribed the extraordinary permanent value of his descriptions and opinions of the mining-districts of Mexico.

The portrait here exhibited shows Humboldt as I knew him, and saw him in Berlin, a few months before his death, which took place in May, 1859. I need not say that I treasure the memory of that interview, as I do the only letter I ever received from him. Both of them exhibited the kindness and sympathy of a great man, crowned with age and fame, towards an obscure youth, fired by his example.

SWEDENBORG.

Emanuel Swedenborg was one of the foremost, if not the very foremost, of all mining engineers and metallurgists who have ever lived. If he had been nothing more, his fame in this respect might have been more exalted and secure. In his versatility, his eager questioning of all natural phenomena, his combination of practical knowledge and skill with theoretical speculations, and his perennial flow of spontaneous suggestions for the benefit of his fellow-men, he resembled our own Benjamin Franklin.

Without attempting to give in this place an adequate discussion of Swedenborg's work, I will merely indicate its nature and scope.

Born at Stockholm in 1688, and receiving his degree as Ph.D. at Upsala in 1709, he started at once on a pilgrimage after

knowledge, practical as well as theoretical. Wherever he went he learned a new trade. First, he became an expert bookbinder; then, as a student in London of the works of Newton, he boarded successively with a watch-maker, a cabinet-maker, and a maker of mathematical instruments, and made himself expert in all these occupations. Wishing to send to Sweden certain geographical globes, which could not be transported in complete form, he learned the art of engraving on copper, and made the plates himself. A little later, having to stay awhile in Leyden, he learned how to grind lenses for telescopes, and acquired a set of the necessary tools.

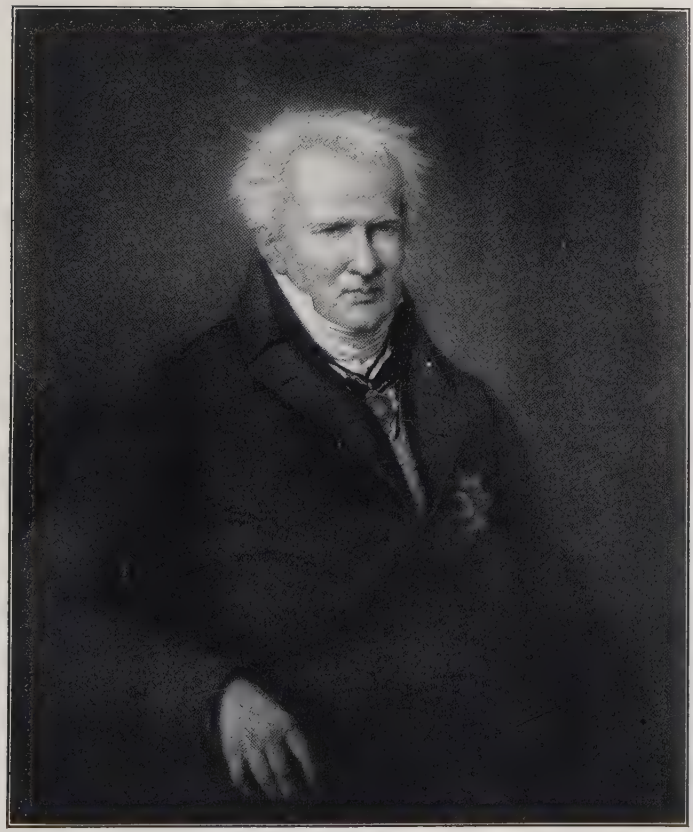
During his stay in England he seems to have been chiefly interested in mathematics and astronomy, and originated, among other things, a method for determining the terrestrial longitude of lunar observations. He occupied himself also with that immemorial problem of astronomical mathematics, the orbit of the moon.

A letter written in 1714 shows that he had turned his attention to mechanics, and had on hand, among other mechanical inventions, the following: for a submarine ship, "which can do great damage to the fleet of the enemy;" a portable siphon for raising large quantities of water, and auxiliary inventions for utilizing this siphon in raising ships, etc.; a machine "driven by fire," "which will put the water in motion" (this sounds like a prophecy of the steam-engine, but was probably something else); new pumps of various designs; new air-guns; a new musical instrument, "by means of which one who is quite unacquainted with music may execute all kinds of airs that are marked on paper by notes" (is this a pianola?); a water-clock; and a flying carriage, showing "the possibility of remaining suspended in the air, and of being conveyed through it."

At this time he was but 27 years old. At 28 he was made an Assessor of the Royal Swedish Board of Mines; but, aside from the routine duties of his office, he was detailed for engineering duty as an assistant of Christopher Polhem, the foremost Swedish engineer of the time. In this capacity he both designed and directed, in 1718, the overland transportation, for about 14 English miles, of a fleet of two galleys, five large boats, and a sloop, as a measure of speedy reinforcement to the King of Sweden in his military operations.

MARTIN LUTHER.

THEODOR KÖRNER.



FRIEDRICH HEINRICH ALEXANDER VON HUMBOLDT.

EMANUEL SWEDENBORG.

In the discharge of his duties as a member of the Board of Mines, Swedenborg labored and traveled incessantly. In 1720, he published his description of Swedish blast-furnaces. In 1723, he presented a memorial, accompanied with complete drawings, urging the manufacture of wrought-iron in Sweden, instead of the shipment of crude pig, to be puddled and rolled in other countries. This proposal, tardily accepted, became the foundation of the manufacture of "Swede's iron," so long a source of profit to that country. He also pressed the home manufacture of salt as a source of revenue; and, having inspected the metallurgical works of Germany and other nations, he recommended improvements in the Swedish manufacture of copper, which would greatly increase its technical economy. Meanwhile, as incidental products of his incessant activity, he published treatises on fire and combustion, and introduced an air-tight stove for the economical warming of Swedish homes. (It will be remembered that Franklin also thought it not unworthy of his genius to invent a stove—which still bears his name.)

Of Swedenborg's *Opera Philosophica et Mineralia*, published about 1733 in three large folio volumes in the Latin language, the second and third volumes were devoted to iron and copper metallurgy, and were reprinted in German and French translations as classics. Dr. John Percy said of them that no books were, in his judgment, "more worthy of the attention of those interested in the history of metallurgy." And an eminent Swedish authority has declared that it is almost impossible to measure the improvement which Swedenborg brought about in the mining and metallurgical practice of Sweden.

In this imperfect sketch, I have ignored the statements, more or less clearly made by Swedenborg, of great doctrines of modern science, such as the circulation of the blood, the atomic theory, the solar origin of the earth and the planets, the undulatory theory of light, the nebular hypothesis, the recognition of heat as a mode of motion, the connection between magnetism and electricity, and the principles of dynamic geology. I cannot further discuss these subjects here, and must content myself with the consideration of one additional question only—namely, how has it happened that Swedenborg's merits as a leader of science and as a professional engineer have been so inadequately recognized?

One reason is, doubtless, that his books and treatises, published in Latin, have not attracted, to the extent which they deserve, the attention of modern students. Another reason may be, that they appeared during a period when the operations of war engrossed the thoughts of mankind. But perhaps the chief reason is, that, in middle life, Swedenborg turned his attention to theology exclusively, and that he sincerely believed himself to be in continual communication with the inhabitants of the spiritual world, and to be receiving from them revelations which it was his duty to communicate to his fellow-men. He had previously written much and well on theological subjects, to which he had always given serious consideration; and the avowed impulse underlying two of his great treatises, on the Blood and the Brain respectively, had been a purpose to hunt through the physical body of man in the hope of discovering the locality and the functions of the soul. But when he began to receive and reproduce celestial revelations, he not only abandoned scientific methods of inquiry in this field, but definitely condemned them, declaring his former work to have been misdirected and useless—as it had by no means been.

The result was, that those who did not accept his revelations were disposed to reject also his scientific observations and theories, following the erroneous psychology, which recognized no sanity whatever in the presence of a hallucination. Some of the partisans of Swedenborg have made the same mistake, arguing that the value of his scientific work warrants the acceptance of his visions of a world unseen. Both were alike wrong. Without pronouncing any opinion whatever as to Swedenborg's theological doctrines or prophecies, I think it is both practicable and just to estimate, altogether apart from them, his earlier works as a mathematician, astronomer, physicist, engineer, and metallurgist; and I do not hesitate to say that such an estimate puts him in the foremost rank.

Biographical Notice of James Duncan Hague.*

BY ROSSITER W. RAYMOND, NEW YORK, N. Y.

(Chattanooga Meeting, October, 1908.)

THE formal outline of Mr. Hague's life and work is embraced in the following statement, chiefly based upon data furnished by him, at my request, shortly before his death. At that time the probability that I would survive to record his death was apparently small; but I urged upon him the duty of providing, whether for me or for my successor as Secretary of the Institute, trustworthy material for the notice which somebody would some day have to write—and which might otherwise be incomplete or incorrect. Mr. Hague, with characteristic promptness and freedom from affectation, did what I asked of him; and I am therefore able to present an authoritative sketch—concerning the details of which I shall add my own comments, and some particulars obtained from friends.

James Duncan Hague was born Feb. 24, 1836, at Boston, Mass., where his father, Rev. William Hague, D.D., an eminent clergyman, then resided. After a preliminary education in private schools, he studied in 1854 at the Lawrence Scientific School of Harvard University; in 1855 and 1856 at the University of Göttingen, Hanover; and from 1856 to 1858 at the Royal Mining Academy of Freiberg, Saxony. In 1859–1861 he was attached as chemist to a South Sea exploring expedition, sent by W. H. Webb, of New York, to examine Jarvis, Baker, and other islands for phosphate-deposits. After the outbreak of the Civil War, he entered the U. S. Navy, serving for a year as Judge Advocate to the fleet stationed at Port Royal, S. C. In 1863 he became Superintendent of the Albany and Boston copper-mine, in the Lake Superior region, where he was also connected with the earliest development of the Calumet and Hecla property. In 1867 he became the First Assistant Geologist in Clarence King's famous Exploration of

* See portrait constituting the frontispiece of this volume.

the Fortieth Parallel, with which he was connected for more than three years, and to the report of which he contributed, in addition to other service, the third volume, treating of mining and metallurgical methods and apparatus. From 1871 to 1878 he practiced as a consulting mining engineer, residing in California. In 1878 he was one of the U. S. Commissioners of the Paris International Exposition of that year, and contributed to the report of the Commission an account of the mineral industries represented in that Exposition. From 1879 until his death he resided (except as to his summer home at Stockbridge, Mass., and his occasional journeys on professional business) in New York City, where he was employed as consulting engineer by many investors and companies, and especially served as the President of the North Star Mines Co., operating at Grass Valley, Cal. Besides the professional publications mentioned above, Mr. Hague was the author of many articles in magazines and other periodicals. He edited in 1904 the Memorial of Clarence King, issued by members of the Century Association of New York, and contributed to that volume one of its most interesting chapters. In 1904 he read before the Mining Engineering Section of the International Congress of Arts and Sciences, held in connection with the St. Louis Exposition, an able paper on Mining Engineering and Mining Law, which was published in the New York *Engineering and Mining Journal* of Oct. 20, 1904.

Mr. Hague was a Fellow of the American Association for the Advancement of Science, and a member of many scientific, business, and social organizations, including the American Geographical Society, the American Academy of Arts and Sciences, the American Numismatic Society, the N. Y. Historical Society, the American Museum of Natural History, the N. Y. Metropolitan Museum of Art, the N. Y. Botanical Garden, the New England Society, the St. Andrew's Society, and the following clubs: the Century, Union League, Metropolitan, City, and Downtown, of New York City; the Union, of Boston; and the Pacific Union, of San Francisco. He became a member of the Institute in 1903, was elected a Vice-President of the Council in 1906, and served for two years as one of the Committee on Membership.

The foregoing summary constitutes, even without comment, an impressive picture of Mr. Hague's honorable and useful career; but I venture to add a few comments from my own recollections and those of others, which may serve to emphasize some features of it.

1. Hague preceded me by a couple of years as a student at Freiberg; but it was my privilege to know, as teachers or as friends, the great men under whom he studied there, among whom I recall especially Cotta, the elder and the younger Weisbach, Breithaupt, Scheerer, Fritsche, and Gaetzschmann. (Richter, afterwards head of the Academy, was then, I think, only a tutor of blow-pipe practice.) These men constituted a combination of theoretical and practical knowledge which has never been equaled in so small a faculty of instruction. Every one of them, through his books and his students, impressed himself indelibly upon his generation and its mining and metallurgical industry; and a man like Hague, passing under their hands, was sure to acquire an all-round training of the highest value—especially if, like Hague, he had had such previous instruction as that which was furnished in those days at the Lawrence School of Harvard and at the University of Göttingen.

2. His career as a mining engineer really began with his work in the Lake Superior copper-region; and I cannot but feel that to him, and others like him, we owe the example, furnished by that region, of the best mine-engineering and mine-management in the United States. Our self-taught mining engineers of the Pacific Coast, struggling with new problems, deprived of trained technical aid, and backed by rich bonanzas, wrought wonders of executive ingenuity and achievement; but their example of technical economy was lamentably misleading to their successors. Afterwards, when labor commanded a smaller, and timber a larger, price; when economies in concentration and transportation had become possible; and when rich ore-bodies had given place to larger masses of low-grade material—it was not to Nevada, but to Lake Superior, that mining engineers could look for guidance; and I do not think I am mistaken in believing that the leadership of the Lake Superior copper-mines in this respect was largely a result of the early work of James D. Hague and others, who had been thoroughly trained abroad.

In connection with this part of Hague's career, I have received from an old member of the Institute, honorably distinguished in later years as the manager of important mining enterprises, a letter narrating his experience when, as a young graduate, he visited the Lake Superior region, with letters of introduction to Mr. Hague, and requested an opportunity to learn the details of mining practice. Mr. Hague received him cordially, and put him, as a workman, through all the operations of rock-sorting, wheeling, loading, mining on day's wages and on contract, etc., yet never forgot to give him special chances for other temporary and instructive work, such as making surveys and maps, setting up machinery, operating the stamp-mill, etc., and also, introducing him to other managers, enabled him to inspect their mines and methods. All this, my correspondent says, was done with an unfailing personal kindness and consideration which he can never forget. I recognize in his story not only the generous character of Mr. Hague, but also the effect of the old Freiberg atmosphere, in which every one, from the professors down to the foremen and common miners of the district, took a friendly interest in the students, and assisted them in every possible way.

Mr. Hague's connection, above mentioned, with the discovery of the Calumet vein, deserves more particular record as an interesting contribution to history. While manager of the Albany and Boston mine, he became intimately acquainted with Edwin J. Hulbert, who often declared to him the expectation of finding a valuable copper-lode at a certain place, and gave the reasons for this belief. When Mr. Hulbert sank the pit which struck the "Calumet breccia," Mr. Hague was one of two or three persons who saw the discovery first. Some of the rock taken from the pit was hauled on sledges to the Albany and Boston mill, where it was treated in the presence of Mr. Hague. In consequence of this test, confirming his previous observations, he secured an option on the tract from which the Calumet & Hecla Co. has since taken many million dollars. But the Boston associates to whom he offered this chance were not willing to expend money in what they deemed a highly hazardous venture; and the option was allowed to expire.

3. Mr. Hague's most important work, and the one which

permanantly established his reputation, was Vol. III. of the Report of the U. S. Geological Exploration of the Fortieth Parallel, entitled "Mining Industry, by James D. Hague, with Geological Contributions by Clarence King," and illustrated with many plates and an accompanying atlas. Of the five volumes which constituted the report of that famous survey, this one (a magnificent quarto of 647 pages) was the first to be published—a circumstance which must be ascribed to the support given by Mr. Hague, through the punctual performance of his part of the work, to the practical wisdom and tact of Mr. King in deciding upon the order in which the reports should be issued.

An instance of the unfortunate consequence of a different order of publication was furnished by the California Geological Survey, for which the Legislature of that State had made several successive large annual appropriations, without seeing "practical" results, in the form of publications recognized as immediately useful to the mining industry of the State. It happened that, when the question of a further appropriation was pending, the only report which had been issued by the Survey was a volume on palæontology; and an opponent of the appropriation carried the House with him by simply reading random extracts from that dry and technical treatise, as samples of the character of the work which had been done at the public expense up to that time. The appropriation was refused; and the valuable work of Bowman and others, on the old river-channels of California and their gold-bearing gravels and cements, was thereby barred from publication for several years. For this result, Prof. J. D. Whitney, the distinguished head of the Survey, has often been blamed, on the ground that he expended money and time in a preliminary topographical and geological survey, without attacking problems of immediate industrial interest. Personally, I think there is some foundation for this criticism. Professor Whitney, with a lofty and serene regard for the logical sequence of science, and an equal disregard for the clamor of industrial interests, had begun his work with the topographical reconnaissance necessary as a basis for accurate geological deductions and correlations; and, in the course of this preliminary labor, his field-parties had made incidentally many interesting palæontological observa-

tions, undoubtedly significant in their bearing upon the geology of the State. Professor Whitney had also started investigations of more immediate and evident practical importance; but unfortunately, in his plan of a permanent and monumental scientific achievement, these were not of prime importance, and were advancing slowly. Probably the thought never occurred to him that it would make any difference what he published first, as the fruit of his work for the State; and thus he made the profound mistake in policy of issuing, merely because it was ready, a learned book on palæontology for the benefit of a limited outside public of specialists, and to the profound dissatisfaction of the people who had paid him, and were, reasonably or unreasonably, expecting something else for their money.

Clarence King, as an employee of the California Geological Survey, appreciated and sought to prevent this error of policy;[1] and probably his views in this respect led to his organization of the Fortieth Parallel Survey. At all events, he took care, in publishing the report of that survey, to issue first the volume which appealed most directly to the imagination and the interests of Congressmen and their constituents; and this piece of tactful management I (as a contemporaneous and disinterested, though friendly, observer), believe to have been, next to King's own persuasive personality, the chief reason of the enthusiastic support which he secured from Congress in the organization, in 1879, of the U. S. Geological Survey, of which, at the age of 37, he was made the first Director.

I recall this story, in order to declare my conviction that the phenomenal career of King was largely determined, at its most critical moment, by the quiet, modest, and thorough work of his chief assistant, James D. Hague, as expressed in the classic Volume III., already mentioned. The other volumes of the same series, dealing with the systematic and descriptive geology, zoölogy, palæontology, and botany of the region covered, could scarcely have aroused to an equal extent the public approval and support.

Moreover, the effect produced by this volume was justified by its intrinsic merits. Clarence King himself contributed to it a description and discussion of the great Comstock Lode,

[1] On this point, see remarks in my Biographical Notice of Clarence King, *Trans.*, xxxiii., 629.

which, although largely a compilation from mine-maps, and an eclectic reconstruction of existing theories of ore-deposition, constituted the first connected consistent and comprehensive statement of the facts and relations involved. The volume comprised also interesting and important descriptions of the geology of certain districts of Nevada by S. F. Emmons and Arnold Hague, and a discussion of the chemistry of the Washoe process by the latter. But the permanent value of all these contributions, I do not hesitate to say, was eclipsed by that of James D. Hague's chapters on mining and metallurgical methods of the Comstock district, and also of central and eastern Nevada and the gold- and silver-districts of Colorado. These chapters not only furnished to students of technical history a trustworthy view of the practice of that period, but also gave the impulse and "set the pace" for the future work of American experts in the same field. They are indeed classics, and while, like other classics, they have been, and will be, complemented by later work, they can never be dethroned from the position conceded to them by universal acclamation upon their first appearance.

I must repeat that this famous Vol. III. did more than anything else could have done to secure for King's subsequent schemes the enthusiastic support of Congress. It proved, without explanation or argument, the practical usefulness of technical reports; and out of it came, as a result, the U. S. Geological Survey, with all its varied and momentous activities.

4. The report of Mr. Hague, as U. S. Commissioner to the Paris Exposition of 1878, on the mining industry of the world, illustrated in that memorable international exhibit, bears the marks of his ability as a trained observer and critic, but does not call for special comment here, especially in view of the fact that the larger part of it was prepared under his direction by his assistant, G. F. Becker. But I have no doubt that Dr. Becker, who has since achieved a professional eminence quite sufficient to warrant him in acknowledging the superiority of another in a particular specialty, would heartily confess the influence which Mr. Hague, as a trained mining engineer, exerted upon his youthful work for this report, thirty-one years ago.

5. While Mr. Hague was engaged in practice as a consulting engineer at San Francisco, it was my fortune to meet him

in court, as an expert witness; and I can cordially testify to the invariable candor and courtesy, as well as the thorough and comprehensive knowledge, which he exhibited in that position.

6. Probably the chief cause of Mr. Hague's practical withdrawal from professional authorship was his acceptance of the Presidency of the North Star Co., operating at Grass Valley, Cal. The famous North Star mine was known to us "old-timers" as one of the most brilliant of those early ventures in gold quartz-mining, the subsequent history of many of which disappointed the sanguine hopes of their promoters. The ore was so rich in gold as to facilitate stealing (now euphemistically called "high-grading") by the miners; the vein was so narrow and so frequently faulted as to incur all the difficulties of underground development and all the legal complexities of that amazing absurdity, the U. S. "law of the apex"; and, with increasing depth, its exploitation encountered also the discouragement of scientific doubt as to the persistence of such deposits below a certain (or, rather, a happily as yet uncertain) level. In the face of these, and of other, possibly still more alarming, difficulties, Mr. Hague successfully managed the property of the company for thirty years, carrying the work forward in times of temporary discouragement by the sheer weight of his superior knowledge and courage; and, according to an opinion expressed by him, not long before his death, he left it with an assured prospect of another thirty years of profitable activity. Regretting, as I cannot but regret, that technical literature was not enriched by further contributions from one whose competency had been so brilliantly displayed, I still hesitate to declare that this object-lesson of wise and effective administration was less important to our profession than the most masterly of monographs.

7. Mr. Hague's literary skill was shown in the Memorial of Clarence King, which he edited, and to which he contributed some of its most interesting pages. Indeed, as a "veteran observer" of similar work, I cannot consent to believe that such skill comes, even to genius, without practice; and therefore I feel sure that Mr. Hague's contributions to current literature must have been more numerous than we can now trace. It used to be said that no marching company of militia could quite equal the easy precision of regular troops; and it may be

said likewise, I fancy, that no author can march like a regular without having somehow become a regular.

8. By reason of his long comparative seclusion from general practice, Mr. Hague did not become a member of the Institute until 1903. In 1906 he was elected a Vice-President, and was appointed to the Committee on Membership. From that time, no member, either of the Council or of the Committee, was more zealous and faithful in the performance of duties essential to the welfare of the Institute, yet not likely to receive even the reward of general recognition and praise.

9. Concerning Mr. Hague's gracious, winning, yet dignified personality, I can hardly trust myself to speak. For he was to me, through many years, a true, congenial, steadfast friend—one of those whose friendship does not need to be nourished by frequent interviews or correspondence, but exhibits itself, after any interval, with unchanged cordiality. I do not believe that such a relation, having already shown itself superior to separations of earthly time and distance, could be severed by the physical accident called death.

Development-Sampling and Ore-Valuation of Gold-Mines.

BY C. BARING HORWOOD, JOHANNESBURG, SOUTH AFRICA, AND MUNGO PARK, GLAN CERIS, NORTH WALES, G. B.

(Chattanooga Meeting, October, 1908.)

THIS paper is intended, in the light of recent investigations, to call attention to some of the essential features of good practice in sampling and mine-valuation. Mine-sampling may be divided broadly into two classes: (1) development-sampling and computation of ore-reserves; and, (2) stope-sampling and estimation of stope-values.

DEVELOPMENT-SAMPLING.

It is with development-sampling only that this paper deals—the term development being taken to mean the opening-up of a mine by cross-cuts, drives, connections, etc., preparatory to actual extraction of the ore for milling. In development-sampling the work of the sampler and the consequent estima-

tion of the probable value of the mine or portions of the mine, as the case may be, cannot be verified until a later period, when the ground thus sampled has been exploited; hence, extreme care and accuracy are essential. Moreover, the development-sampler has to foresee and allow for contingencies which may arise when stoping is started. It follows that the sampling, in development-work, demands familiarity with the conditions likely to be encountered later in actual mining. In stopes, on the other hand, sampling is done under actual working-conditions, and the sampler's valuations are immediately checked by the gold-output, or in some cases by a second sampling of the broken ore. A stope-sampler therefore need not be an experienced miner, though he should possess sound technical training and work accurately. Should he be somewhat inexperienced, he soon has his errors discovered for him, and learns to avoid them. Taking these facts into account, it is obvious that a mining company employing two or more samplers should choose for the sampling of development-work the man who is most experienced in the general and local conditions of mining.

The earliest stages of mine-development often consist of shaft-sinking and cross-cutting. If the shaft be sunk on the lode, ore-sampling of course commences at once. In any case, in a vertical shaft particular attention to the formation should be paid by the person in charge of the sinking. We know of a recently-sunk vertical shaft in which the reef was passed through unnoticed, and much money and valuable time were wasted in sinking to a considerable depth below the reef before it was discovered. Detailed information of the ground passed through, accompanied by illustrative specimens, should be preserved, although frequently too little attention is paid to this matter. It is usually the duty of the person in charge of development-sampling to note all dislocations, faults, and dikes as they occur, and this work cannot be done efficiently without a knowledge of the local formations. At times the condition of the country-rock can only be determined by shafts and cross-cuts. A strict watch of the conditions as they develop may sometimes result in the discovery of a fresh payable deposit. Mining history is replete with many instances of adventitious finds of a mineral or vein resulting while searching for others.

A remarkable instance of recent date was the discovery on the Rand of a pyritic seam in the quartzites behind the main reef that, in places, was of economic value. During the past 20 years that this seam has been known its importance was not appreciated. On some mines on the Rand, in cross-cutting, every foot is sampled and assayed as the work proceeds, and under this careful method of working no auriferous rock of any kind is likely to be overlooked. Cross-cuts should be examined, not merely on one side, but on both. It is very important that information gained during the early progress of a mine should be recorded for reference, since too often the geological story told in the early stages of development is lost, the unrecorded testimony being hidden in lagged-up shafts and cross-cuts.

Development-sampling should proceed *pari passu* with the development-work; if the former lags money is too often wasted on work in unprofitable ground.

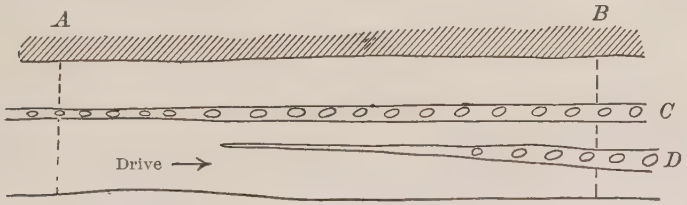


FIG. 1.—SKETCH OF SECTION OF AN ORE-BODY.

An important point in the exploration of an ore-body is that it should be continuously examined throughout its entire course. By taking samples at occasional points only, the presence of intervening irregularities, horses, forks, or branches, may be overlooked.

Attention should also be paid to the formation outside of the pay-streak, so that new ore-bodies will not be passed unnoticed. Fig. 1 represents an occurrence not uncommon on the Rand—namely, a reef, *D*, which has "made" under reef *C*, between the points *A* and *B*. The examples above given may seem to some to belong to the rudiments of practical mining, but the precautions mentioned are so often neglected that we do not apologize for emphasizing them here.

With regard to the taking and ticketing of samples, numerous ideas have been suggested and adopted for this work, vary-

ing with the requirements of particular circumstances. So much has been written on the subject that it is unnecessary to deal with it at length in this paper. It may, however, be as well to remark that in taking a sample the bag or other receptacle should be clearly numbered, or else a numbered ticket be inserted. The latter method is preferable if there is any danger of salting. All details of each sample should be entered with the reference number in the sampler's notebook, on the spot, at the actual time of taking the sample, and should generally be accompanied by a sketch-section, also made at the time, giving the position relative to some known point.

The results obtained by sampling should be carefully recorded for reference. It is by no means common to find records that have been well kept. At one mine the sampling results are filed carelessly on slips of paper; at another, a bewildering array of columns and figures are inclosed in a library of leather-bound volumes. In such books are specified the date of taking the sample, the number of the level or stope, as well as of the sample, the sampler's name, the thickness of reefs, the thickness of waste, the total thickness of reef, the total thickness of waste, and so on *ad infinitum*—in fact, so much indigestible detail that the main issues are obscured.

In both of the cases above described any required information is readily obtained if those responsible for the records are at hand. What is wanted, however, is a record that explains itself, just as a merchant's book shows clearly the nature of that merchant's business to an examining auditor.

In a mine in which numerous samples are taken daily, it is desirable that particulars of these samples should be recorded at once, in order that they be not lost or forgotten, and that the assays should be done and the results entered up as soon as possible, so that the latest sample-details are available at the earliest possible moment for those whose business it is to know them. For the purpose of recording these particulars a sort of sample day-book should be kept in the office, in which is entered the sampler's name, the date, the number of the sample, and such details with regard to the latter as may have been obtained. This day-book should have the pages numbered so that it can be referred to in the stope- or development-sample record-

books. Postings, of essential summaries only, to the latter books should be made from time to time. By this system the record-books are not obscured by unnecessary details, while further information, if required, can be obtained at once by referring to the day-book. The data in the day-book would, of course, be arranged chronologically, and the record-books should have as their basis of reference the locality from which the samples were derived.

In sampling-memoranda, distances should be referred to common starting-points when possible. This is not always done, and it is by no means rare to find books in which one sample-section is described as being 13 ft. east of, say, peg 25 W, while the next section is measured from some other peg, or even from a pillar. However lucid such observations may be to the sampler, they convey little information to others.

At the foot of every page the number of sections and the totals of such columns as can be added should be entered. These totals will be found to save much work in estimating the ore-reserves.

Clearness and simplicity should be a characteristic of the records thus described. At some mines a special clerk, known as the sampler's clerk, attends to the records, thus enabling the samplers to devote more attention to the careful taking of samples.

Pictorial information is, perhaps, best supplied to the records by means of the continuous-section method, an excellent system that has been well described by M. H. Burnham.[1] This method of illustration shows readily the position of the pay-rock in a drive or stope-face. The drawing need not be so elaborate as to preclude its insertion in the record-book alongside of the written data.

Record-books, however excellent, are not the best means of conveying geological information, and it is not easy to picture the distribution of ore-shoots by their aid. This is best done by means of the assay-plan.

The object of the assay-plan is largely to separate the sheep from the goats—to point out payable and unpayable zones. The assay-plan generally conveys most information when made

[1] Continuous Section System Mine Sampling, by M. H. Burnham, *Transactions of the Institution of Mining and Metallurgy*, vol. x., pp. 204 to 211 (1901–2).

on the plane of the reef. On the Rand very neat assay-plans are made, though in many mines there is a tendency to crowd the plans with individual assays. Payable blocks should be tinted with a slight wash of distinctive color. While in some mines there is a prejudice against indicating values by tints, other mines, on the contrary, have plans resembling Joseph's coat, each grade having its own special shade. The latter extreme seems unnecessary, since the figures should tell their own tale, the distinction between "payable" and "unpayable" being usually a sufficiently graphic illustration.

As soon as ground has been blocked out and included in ore-reserves, the value of these blocks should be plainly indicated on the assay-plans. All faults and dikes should be shown on the plan, and it is sometimes advisable to indicate important changes of the foot- or hanging-wall.

The computation of average value from assays has been thoroughly discussed recently by Mr. Crosley, T. A. Rickard, and others.[2] That the somewhat crude methods of averaging sample-values at present in vogue are not theoretically precise has been proved beyond doubt. At the same time the exact rules governing the deposition of ores remain unknown. Let it also be remembered that the present "inch-dwt." method of taking a geometrical mean has been proved to yield an excellent approximation in averaging stope-values. Some of the reasons why faulty estimations of ore-reserves are so frequent will be referred to later. The present geometric-mean method is also very widely used, and to abolish it would be to condemn the method by which most of the world's gold-mines have been valued. As the case now stands, the exact method of averaging sample-values yet remains to be discovered. All known methods yield only approximate results, which is not surprising when one reflects that each sample often represents more than 100 tons of ore blocked out. We therefore prefer to use for the

[2] The Computation of Assay Values, by William Crosley, with discussions, by A. G. Charleton, Dr. T. Kirke Rose, S. J. Speak, L. H. Cooke, A. H. Ackermann, Reginald W. Robinson, and Wm. Frecheville, *Transactions of the Institution of Mining and Metallurgy*, vol. xv., pp. 89 to 125 (1905–6).

We call special attention to the very useful book entitled, *The Sampling and Estimation of Ore in a Mine* (Hill Publishing Co., New York, N. Y.), compiled by T. A. Rickard from the *Engineering and Mining Journal*, and containing articles by many leading American mining engineers.

present the simple "inch-dwt." method, which, after wide usage, has proved fairly satisfactory, to other methods, which are more cumbersome, less used, and which have not yet been shown by practice to give a greater approach to accuracy.

The problem of dealing with abnormally high assays is of lasting interest. The general consensus of opinion is that abnormal values should not be retained. From experience we consider the best plan is to obtain the average value including the abnormal assays, and substitute the value thus obtained for the abnormal one, and again average. Some definite method, such as the above, of dealing with abnormally high assays might well be adopted at every mine. The practice of rejecting a high

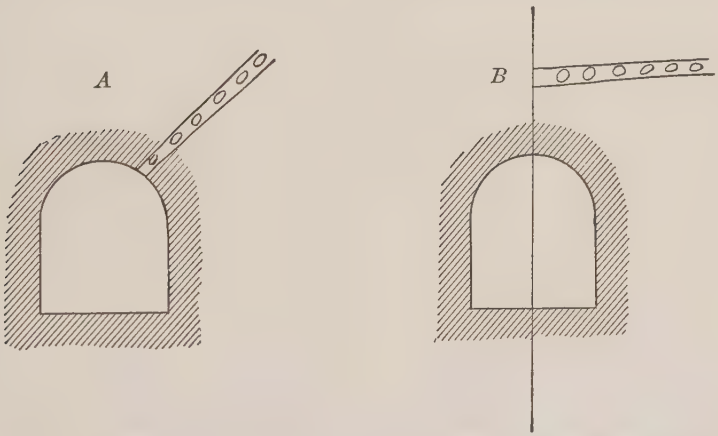


FIG. 2.—SECTIONS OF ORE-BODY SHOWING CONDITIONS AFFECTING COST OF MINING.

assay in order to insert a haphazard low one is not satisfactory. In every case the method followed should be stated.

In the case of missing sections, such as a section where it is impracticable to sample the reef, but where the reef is known to exist, we suggest that the missing section should be taken (having regard to the known presence or absence of ore-shoots or local enrichment and allowing accordingly) to be the mean of sections of equal length on either side, such length, unless the special circumstances of the case demand otherwise, being at least equal to the length of the gap. Supposing that it was impossible to sample a drive for 20 ft., the value of that 20-ft. length would be taken to be the mean value of the two

20-ft. lengths bounding it on either side, provided always that the reef is shown to exist in the section.

When one of the adjacent sections is an exceptionally rich ore-shoot, it is obviously impossible to apply any hard and fast rule.

Ore-Valuation.

In the valuation of the ground developed, the ore should be divided into blocks of "payable" and "unpayable" value.

The actual grade which is to be classed as payable depends upon the circumstances existing in the mine itself and the working-conditions of the district. Local conditions in the mine will sometimes vary materially. For instance, in a district in which working-costs average about 17s. 6d. a ton, a 3.75-oz. 2-in. reef may be payable in one case and not in another, as is shown in Fig. 2, *A* and *B*. In the former, *A*, a comparatively narrow (say 30-in.) stope could be carried and the ore would gravitate readily into the level; in the latter, *B*, owing to the flatness of the pitch, a large amount of waste would probably have to be mined with the reef, and much labor be expended in transporting the broken ore. The above example emphasizes the fact that it is often desirable to use different factors for different blocks, instead of taking a constant factor of payability for the whole mine. It is not always easy to draw a sharp line of demarcation and to divide all the developed ground into payable and unpayable blocks. Sometimes several high assays occur together in poor tracts of ground, and sometimes bunches of poor ores are found in rich blocks.

In dividing the payable ground into blocks consideration should be given to the placing of future stopes, so that the management will know readily how to attack the ground from the various working-faces in order to maintain a certain grade of ore. In estimating the grade likely to be obtained from a block of ground which is traversed by dislocations, it must be remembered that in carrying a stope through normal faults much waste may be mined, which will lower the grade. This fact is not always borne in mind, since many computators deduct a percentage from the tonnage for faults, without altering the grade. In practice the tonnage is usually increased in such a case owing to the waste that is mined along the break, and the grade is thereby impoverished. On the other hand, it may sometimes

be possible to improve the grade (and lower the tonnage) by arranging pillars in the poorest ground.

One of the most important questions in ore-valuation on narrow deposits is that of the probable average stoping-width. On the Rand assumptions of width have almost invariably been too narrow. In one case, a stoping-width of 30 in. was assumed where 2-ft. of the hanging-wall a little above the reef was so loose that it would obviously have to be broken with the latter. It is hard to understand why a desire to minimize the stoping-width should so often be found. Owing to this practice, a mine which has been rated by development-sampling as, say, a 12-dwt. proposition, too often turns out 9-dwt. rock. Moreover, in the case of certain mines the mining-costs are burdened at the

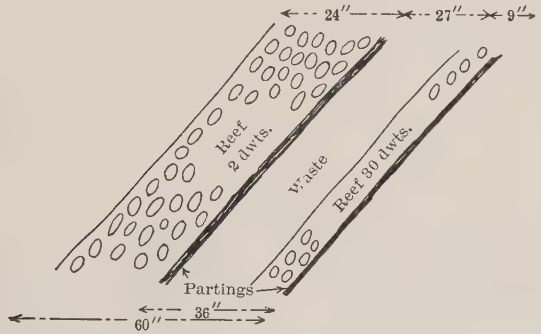


Fig. 3.—Section of Ore-Body Showing Two Widths of Working.

outset with a higher charge per ton for development-redemption than necessary, owing to the lower tonnage estimated on the basis of an impracticably narrow stope-width.

It is perhaps only just to add that in some cases the estimated width was probably sufficient, but the stopes were subsequently carried unnecessarily wide owing to careless mining. In assuming a stoping-width, not only the minimum to which the ground can be made to break must be considered, but also the amount that may subsequently come away as the face advances. This latter amount is frequently considerable, and unless it is used for filling it generally finds its way to the mill sooner or later. Mr. Way[3] estimated the amount of this waste material at 12 in., but it is inadvisable to assume any arbitrary figure

[3] Ore Valuation of a Witwatersrand Mine, by E. J. Way, *Transactions of the Institution of Mining and Metallurgy*, vol. xv., pp. 134 to 136 (1905–6).

for all cases, since the scaling will be negligible in some cases and considerable in others; and it will also differ in hand- and machine-stopes, being naturally greater in the latter.

Concerning the amount of ore likely to be sorted out, this question should be considered in the case of each block when assuming its milling-width. The too-common practice of assuming one uniform stoping-width and a constant sorting-factor for the whole mine, regardless of the variations of dip, nature of the country-rock, or reef thickness, is to be deplored.

In the case of very wide reefs it is necessary to decide whether it will not be more profitable to work only a portion instead of stoping out the whole width. Fig. 3 illustrates a case in which a choice is offered between mining 36 in. of 7.5-dwt. ore or 60 in. of 5.3-dwt. ore.

Problems like those referred to in this paper have to be considered before ore-reserves are calculated, and they demand for their solution a thorough knowledge of mining conditions acquired by sound technical training coupled with wide practical experience.

In conclusion, no hard and fast rules are advisable for ore-valuation, except in one or two cases, and in the main these rules are inappropriate for trained mining engineers. However, it is desirable that in formulating hypotheses reason and method should be observed, and that assumptions, when made, should always be stated, so that opinions may be formed of their value.

Gayley's Invention of the Dry Blast.

BY R. W. RAYMOND, NEW YORK, N. Y.

(Chattanooga Meeting, October, 1908.)

THE immense commercial value of the Gayley dry-blast process has been established beyond controversy. The testimony of practical blast-furnace managers, on both sides of the Atlantic, agrees that it reduces the cost of pig-iron about $1 per ton; but from the same testimony it appears that this direct economy does not by any means cover its commercial advantages—in fact, that under conditions frequently encountered in practice, it presents other benefits of even greater immediate financial importance. To this point I shall recur in a later part of the present paper.

The history of the reception given to this invention, especially on the other side of the Atlantic, has repeated an experience familiar to American engineers. First, the reports of our practice were rejected as theoretically impossible, according to generally accepted notions and formulas; then the figures were reluctantly accepted; and finally the attempt has been made to explain how they could have been obtained. As to the Gayley dry blast, we Americans have been for several years in the last stage of the discussion, and the rest of the professional world is rapidly coming to it.

Meanwhile, the remark has been repeatedly made that, whatever Mr. Gayley's process may turn out to be worth, it was not an invention on his part, but simply the energetic and persevering execution of a procedure already proposed by others. This question I wish to discuss first in the present paper.

I. WHAT DID MR. GAYLEY INVENT?

Realizing that this is not the place for detailed argument on an issue of patent law, I shall not undertake to quote and criticise the publications prior to Mr. Gayley's first patents. It may be admitted, without such critical analysis, that the possible advantages of removing moisture from the air forced into an iron blast-furnace were recognized and talked about a hun-

dred years ago, and again fifty years ago; that methods and apparatus for cooling air (and incidentally precipitating moisture from it) were known before Mr. Gayley's invention, and that, in one case at least, a system of cooling the blast for a converter had been actually patented. All these prior prophecies and proposals might be shown in detail to be impracticable, defective, or visionary; but it is not my present purpose to discuss them. The fact remains that, after they had all died out, to the last echo, and after the object they sought had been pronounced by high scientific authority, with the general consent of technical experts, economically unattainable, Mr. Gayley went ahead, and did the thing!

This alone might not be conclusive as to his claims, since mere energy and perseverance, however meritorious, do not constitute invention. But this is not all. The significant fact must be added that, after beginning, with all the light that previous investigation and invention could give him, his attempt to carry out the "well-known" operation, Mr. Gayley spent six years in costly experiments before he found out how to do it. This certainly raises a strong presumption that during those six years Mr. Gayley discovered something not previously known; and it is that discovery which I wish to define.

All previous plans or suggestions for removing moisture from the blast by reducing its temperature have two features in common.

1. Following the analogy furnished by the natural precipitation of dew, they assume that moisture can be effectively removed from the blast in proportion to the reduction of its temperature to successively lower dew-points.

2. They assume that this operation would be metallurgically advantageous in proportion to the extent to which moisture was thus removed. Considering the objectionable effect of such moisture to be a direct function of its amount, they all infer that removing part of the moisture would, to that extent, remove part of the evil, and that the benefit thus realized would be positive, even if it were but partial.

Technical Efficiency.

The first of these assumptions is based upon an incomplete conception of the analogy of nature. Closer observation should

have shown these theoretical inventors that the lowering of the temperature of the atmosphere below its dew-point does not produce the deposition of dew, except in a quiet atmosphere, because wind prevents the fall of dew and drives it away as fog. Moreover, all the inventions for taking moisture out of air by cooling it deal with bodies of air practically at rest. Nobody before Mr. Gayley had attacked the problem of taking the moisture out of a hurricane in that way; and everybody who had talked, however vaguely, of doing something of the kind with the blast for a furnace had overlooked the circumstance that simply segregating the moisture by such means, as a mist, and then letting the mist be blown into the furnace, would secure no metallurgical benefit whatever.

By years of patient experiment, Mr. Gayley discovered, and was the first to declare, that if the reduction of temperature were carried to or below the freezing-point of water, the ice or snow thus formed could be precipitated as well as segregated; that (with proper proportions of apparatus) it could be practically caught and held, so as to permit a blast free from suspended mist or fog to go forward into the furnace. The amount of aqueous vapor still held in the air would depend, of course, upon its temperature. There would be less of it at 10° below freezing-point than at that point. But this small difference has little practical significance. The main point is, that unless the moisture be not only segregated by cooling, but actually solidified, its approximately complete deposition from a rapidly-moving current cannot be effected. Practically, therefore, the effect of going below the freezing-point is to make sure that that point has been effectively reached in all parts of the air-current.

Technical Certainty.

With regard to the second assumption above stated, it follows from what has been said already that no reliable metallurgical advantage can be secured by cooling the blast to any temperature short of the freezing-point. For, under such circumstances, not only might moisture thus segregated be carried onward as mist, but the possible precipitation of a part of it as rain, *en route*, would introduce a new and serious evil—the evil of uncertainty. This point deserves special discussion.

The significance of certainty as a necessary element of any

real industrial art has been often overlooked. To such loose logic we owe much fascinating but misleading rhetorical celebration of the "lost arts" of the ancients. In spite of the eloquence with which these "lost arts" have been extolled by orators and amateurs, I have been brought by much patient study to the conviction that (as a general rule, at least) no real arts have been lost. For an art is a process which arrives with reasonable certainty by the intelligent use of definite means at a desired and foreseen result. This proposition may be made clearer by illustration. For instance:

Certain ancient tribes are said to have heated iron-ore and wood or charcoal together in rude hearths, with the aid of the wind blowing across a mountain-top, or of a primitive bellows, supplying an artificial wind, producing thus a half-fused conglomerate, out of which could be selected pieces of true steel —indeed, of tool-steel or razor-steel, of exceptionally high quality. But the assertion that these ancient experimenters had the art of making steel is not justified by any such stories, however authentic. To combine blindly the materials and forces furnished by nature, and then to overhaul the result, seeking for fortunately valuable products, is not to practice an art. Such vague experimenting may be the beginning, out of which, with greater knowledge and the growth of conscious skill, an art may come; but an art it is not. For an art demands the intelligent use of definite means for a definite end.

On the other hand, it is not easy to say at what point a primitive and empirical procedure attains the rank of an art; and it is not necessary for my present purpose to fix that point. The much more important proposition which I wish to emphasize is this: that every industrial process involves the two elements of ignorant dependence upon natural conditions and intelligent control of such conditions; and that the degree of perfection attained by a given art is the degree in which the former element has been superseded by the latter. In practice we try to make a specified product by a given process; yet often, in spite of all our scientific or traditional precautions, a part of our output fails to meet the specifications imposed upon us, and must be rejected; so that, in a general way, it is fair to say that the perfection of an art is measured in inverse ratio to the proportion of such "rejections."

The progress of our metallurgical arts towards perfection in this respect has been largely due to the exact determination of the composition of our raw materials. We can all remember when American iron- and steel-works began to employ chemists of their own, and to determine by chemical analysis not only the character of their products, but also the nature of their ores and fluxes. The result has been a revolutionary transformation of our blast-furnace practice. The old "founder," who diagnosed his slag, and dosed his furnace accordingly, has disappeared. The superstitious reliance upon this or that brand of ore as a material for this or that brand of iron has gone with him. The chemist dictates the charges, and tests the product.

Yet, with all this increase of analytical control the blast-furnace has continued to be, as a great authority once described it, at once the crudest and the most delicate of metallurgical apparatus, subject to inexplicable irregularities, and seemingly obeying whims of its own, beyond the prevision or regulation of the most scientific manager.

One reason (and, in my judgment, the chief reason) is, that while we have analyzed ores, fuels, and fluxes to the second or third place of decimals, we have practically ignored the composition of the blast which goes into the furnace, although this material constitutes more than half the total weight of the materials charged. This defect in control of the furnace process has not escaped notice. The effect of varying moisture in the blast has been recognized for a century past, especially as between summer and winter; and furnace-managers have adjusted burden and blast, in a rude way, to meet the conditions thus created. But I do not think that the significance of diurnal and even hourly variations of the moisture in the blast was ever fully recognized until Mr. Gayley called attention to it. At all events, he was the first to propose a practical remedy for the evils resulting from such frequent changes.

II. Theoretical Tests.

The nature of these evils deserves here a preliminary word, especially because it affects profoundly the validity of all our theoretical calculations of heat-economy, etc.

The influence of moisture in the blast may be summarized with sufficient accuracy for my present purpose, as consisting

of two elements: (1) the effect of a useless constituent, diluting the effective oxygen of the air, and absorbing heat in acquiring the temperature of the other materials; and (2) the effect of a constituent which, by its dissociation, absorbs heat in the hearth (where heat is most needed), and either, by the recombination of its elements, restores that heat in another part of the furnace (where it is neither needed nor desired) or else, through incomplete recombination (evidenced by the presence of free hydrogen in the furnace gas), fails to restore a part of the heat it has absorbed.

We are accustomed to determine the economy of the blast-furnace process by means of thermo-chemical equations and heat-balances, in which we take account of the composition, temperature, specific heat, heat of combination or separation, etc., of the materials entering the furnace, and the composition and temperature of the materials escaping from it. With due allowance for the incidental loss of heat by conduction and radiation, this method should be accurate, if the data upon which it rests are accurately determined. That it fails to give us a perfect criterion of our practice is due, in my judgment, to the circumstance that it is necessarily based on averages, and assumes these averages to represent uniform conditions: such and such a quantity, pressure, and temperature of blast, composition of ore and flux, and quantity and grade of pig-iron produced. These data are usually averaged from considerable periods; in fact, the longer the period taken, the stronger the assumption of the trustworthy character of the calculation—an excellent rule for most purposes, since averages eliminate incidental variations. But when incidental variations are directly influential upon technical and commercial economy, it is not to be hastily assumed that they can counteract one another, so that the average result is equivalent to that of uniform conditions. Suppose, for instance, that a certain temporary change in one factor of the process would have an injurious technical or economical effect; and that a subsequent equal change in the contrary direction would likewise have an injurious effect. Evidently, the net result would not be fairly represented by calculations based upon the average conditions of the period embracing both changes, and upon the assumption that they canceled and neutralized each other.

That this supposed case is not imaginary, I shall try to show. But first I would point out another defect in our usual method of stating the "heat-balance" of the blast-furnace. Our estimates of the utilization and the waste of heat may be practically fair enough, so far as the heat-requirements of reduction and fusion, and the loss of heat in slag and gases are concerned; but they are seldom based on accurate data as to the grade of the pig-iron actually produced. At least, so far as I now recall, the highest degree of accuracy in that respect does not go beyond a recognition of the general distinction between different grades, from foundry to white iron, or of the special heat-requirements of pig-iron of peculiar chemical composition (as to silica, sulphur, manganese, etc.). . The statement that a given furnace is "running on" this or that grade of iron, and the use in calculation of the figures appropriate to that grade, seem to be the best that has been achieved in this regard. But a furnace "running on" a particular kind or grade of iron not only may, but in present practice invariably does, produce more or less iron of other grades (sometimes colloquially called "off-iron"), the amount of which, sometimes even exceeding 50 per cent. of the total product, may seriously affect the value of our technical calculations of heat-economy, if these do not include it as a factor. I offer this suggestion as a partial explanation of the fact that such theoretical calculations do not always furnish a safe criterion of alleged or possible technical economies. The method is scientifically sound; but it is applied to data too roughly determined for such precise mathematical discussion. Many of us have heard of the distinguished engineer, of whom it was said that in "duty-trials" of engines he "would carry out to the third decimal place the determination of the weight of the ashes, while he guessed at the amount of coal shoveled into the fire-box." Possibly some such inconsistency may explain the hasty conclusion of some foreign experts that the reported technical economy of Mr. Gayley's process was "simply impossible."

III. Commercial Economy.

As regards commercial economy, on the other hand, there is no room for doubt or contradiction. If a blast-furnace is "running on" (*i.e.*, managed with the purpose of producing)

a particular kind of pig-iron, and if Mr. Gayley's process will deliver it altogether, or to an unprecedented degree, from the risk of producing incidentally another kind, not called for, and probably not desired or readily salable at a profit, the commercial value of this insurance is beyond measurement by any technical formula that has been or could be constructed. The case presented in Mr. Cook's paper, Experience with the Gayley Dry Blast at the Warwick Furnaces, Pottstown, Pa.,[1] furnishes a striking illustration of this proposition. As already observed, the testimony from iron-works both in the United States and abroad agrees in declaring that the Gayley process reduces the cost of pig-iron about $1 per ton; but this saving, though important, is trivial compared with the commercial advantage of a more effective control of the operation and product of the furnace. To state the case roughly, the Warwick Iron & Steel Co. was caught, with innumerable others, in the financial revulsion of 1907, which stopped for a time the market-demand for pig-iron. It had a profitable contract with solvent customers for iron of a special grade; but the old and almost dilapidated furnace, which it had kept in blast for the purposes of that contract, was running so irregularly that only half—or less—of its product could be delivered under the contract, and the rest would have to be stored as not immediately salable, and, indeed, as never likely to be salable at a price covering the special expenses incurred for the purpose of producing the special and more costly grade of product for which the furnace had been burdened and operated. Under these circumstances, the interest on the capital represented by the "off-iron" would have exceeded the profits on the proportion of special iron deliverable under the specifications of the contract; and a prudent manager would have been obliged to accept the unwelcome alternative (adopted, in fact, by most of our American merchant furnaces) of sacrificing his pending contract, blowing out his furnace, and submitting to the losses in general expenses, interest, etc., and the even greater damage caused by the scattering of skilled and trusted workmen, and the inability to take immediate advantage of a general revival of business, or of a sudden special opportunity for a local resumption of work. These disastrous effects of a suspension of operations are, as I

[1] P. 705, this volume.

need scarcely say, those most dreaded by technical managers, since, besides their direct financial results, they involve the immeasurable anxiety and responsibility of subsequent reorganization.

Fortunately for the Warwick company, it had just completed the installation of an expensive plant for the Gayley process; and the operation of this plant, under all the disadvantages of new and untried apparatus, inexperience of both manager and workmen, and dilapidated condition of the furnace then in blast, enabled the company to raise the proportion of its immediately and profitably marketable product from below 50 to above 80 per cent.; to fill its pending contract; and to realize, instead of industrial demoralization and financial loss, a substantial profit from continued operations. Indeed, it is no secret that, through the total gains of this campaign, the entire cost of the installation of the Gayley system, including the sum paid for the patent right, was repaid in a few months, though the certified saving of $1 per ton in the average cost of pig-iron would not by any means have accomplished that result in so short a time.

The situation above described is one which any manager of a "merchant" blast-furnace (*i.e.*, a furnace selling its product to outside customers) may at any time encounter. But it carries a meaning also for establishments, like the works connected with the U. S. Steel Corporation and other great concerns, which have a use for the "off-iron" produced by their blast-furnaces. For such "off-iron" could be manufactured at smaller expense than when it is turned out as an unwelcome by-product from a furnace charged and operated, at extra cost, to yield a more valuable product. In short, every blast-furnace manager knows that both technical and commercial economy, as well as relief from personal anxiety, would be secured if he could only be sure of making what he is trying to make.

IV. What Has Mr. Gayley Done?

This brings us back to the inquiry, what is the bearing of Mr. Gayley's invention upon this desirable element of certainty, through complete and intelligent control, in the operation of the blast-furnace?

As I have already observed, we have sought to secure such

certainty through the minute analyses of all the raw materials, etc., except the air of the blast, which weighs more than all the rest put together. But when we come to consider this element, we perceive at once that it cannot be usefully analyzed like ore, flux, or fuel. We cannot determine its composition and then store it until we wish to use it; and if we could analyze it as it enters our blowing-engines, our knowledge would come too late to permit any effective action on our part, based upon such information. If we would attain that certainty of control which constitutes the perfection of an art, we cannot treat the air of the blast as we do all the other elements of the charge, which we regard as variables, to the character of which we adjust our practice. The air, most variable of all, cannot be thus dealt with. We must make it practically a constant. Mr. Gayley has shown us that the only way to do this is to freeze the moisture out of it. No attempts at vague amelioration by partial measures will meet the case. What we want is, first, to know just what we are putting into the furnace through the blast; and, secondly, how we can continue to do that particular thing practically without variation.

Mr. Gayley, after years of costly experiment, has shown us, for the first time, how to attain this object; and I am not surprised that leading iron-masters, in this country and abroad, have recognized this invention as the greatest advance in blast-furnace practice since the introduction of the hot blast by Neilson. It is scarcely too much to say that this invention, completing our mastery of conditions previously uncontrollable, has elevated the manufacture of pig-iron from the category of processes which are partly art and partly accident to that of the true arts, which may be practiced with approximate scientific certainty, and therefore with complete confidence of commercial success.

Experience with the Gayley Dry Blast at the Warwick Furnaces, Pottstown, Pa.

BY EDWARD B. COOK, POTTSTOWN, PA.

(Chattanooga Meeting, October, 1908.)

Introduction.

The installation of the Gayley dry-air process appealed specially to the management of the Warwick Iron & Steel Co., for the reason that for fifteen years records had been kept at the works of the company, showing the amount of water entering the furnace in the form of aqueous vapor, and careful observations had been made of its effect upon the working of the furnace. Some years had proved much worse in their effect than others, on account, not only of excessive moisture, but also of great variation in the moisture from day to day, and sometimes within a few hours. It had been found impossible to make a satisfactory percentage of high-silicon foundry-iron in summer, even on high fuel; and for some years the endeavor had been made so to arrange the sales that, during July and August, only a small percentage of the iron produced would be required to carry more than 2 per cent. of silicon. Our larger furnace, running on basic iron, year after year, produced in February a tonnage 20 per cent. higher than in August, on a much lower fuel-consumption, while, at the same time, the summer months were marked by "messes," tuyeres closed by slips, and other irregularities, constituting serious additional business losses.

It will be easily understood, therefore, that the records made at the Isabella furnace after the installation of the dry blast in 1904 possessed special interest, and were closely watched by us. On the basis of our own experience, we argued that if the moisture could be kept low, and also uniform, the cost of installation would be more than justified. The subject was brought before our Board of Directors in the fall of 1905; and in the spring of 1906 it was decided to install a plant of sufficient

capacity to treat 70,000 cu. ft. of air per min., an amount much larger than that treated at the Isabella plant, and sufficient for the maximum requirements of our No. 1 and No. 2 furnaces. It was not until August, 1907, that our plant was ready for operation, and then only two of the five compressors had been finished and installed.

Since no description of a Gayley dry-blast plant, except that of the Isabella furnace, has been presented to the Institute, a brief description of the Warwick plant may be of interest.

The form in which Mr. Gayley's invention was applied at our works may be outlined as follows: Ammonia, liquefied by pressure, is allowed to expand in pipes, inclosing smaller pipes which carry brine; and this brine, thus cooled to a point below 0° C., is conducted through a refrigerating-chamber in coils of pipe, over which the air of the blast for the furnace passes, and upon which it deposits its moisture before entering the blowing-engine. The general principle of the arrangements is the same as that followed in the Isabella apparatus, and described to the Institute in Mr. Gayley's papers;[1] consequently, only the differences in machinery and construction exhibited in the Warwick plant will be specially noted here.

The refrigerating part of our plant, which was furnished by the York Manufacturing Co., of York, Pa., comprises five vertical single-acting compressors, shown in Fig. 1, each employing about 220 h.p., and having 175 tons' refrigerating-capacity (that is, capable, according to the estimate of the manufacturers, of melting 175 tons of ice in 24 hr. of continuous operation. The heat absorbed in liquefying a ton of ice is 284,000 B.t.u.). The object in having so many units is to guard against serious results from break-downs, and also to have a more adaptable apparatus. It is only in the summer months that the total capacity is required. In winter, one compressor will do the work for one furnace. An atmospheric condenser was installed above each compressor.

The brine-coolers are of the double-pipe class, with 3-in. outer pipes, in which the ammonia expands, and 2-in. inner pipes, through which the brine passes in the opposite direction. Through these coolers, and thence through the coils in the

[1] *Trans.*, xxxv., 746 (1905); xxxvi., 315 (1906).

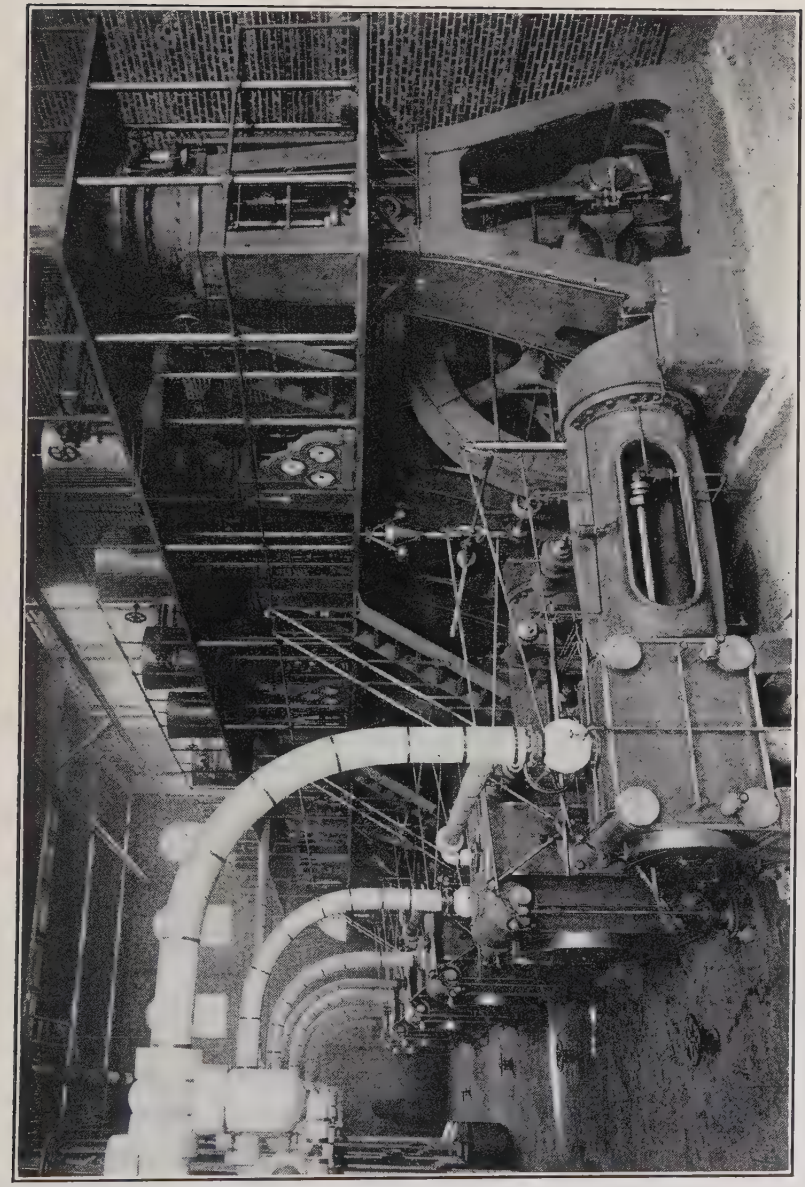


FIG. 1.—VERTICAL SINGLE-ACTING AMMONIA-COMPRESSORS AT THE WARWICK FURNACES, POTTSTOWN, PA.

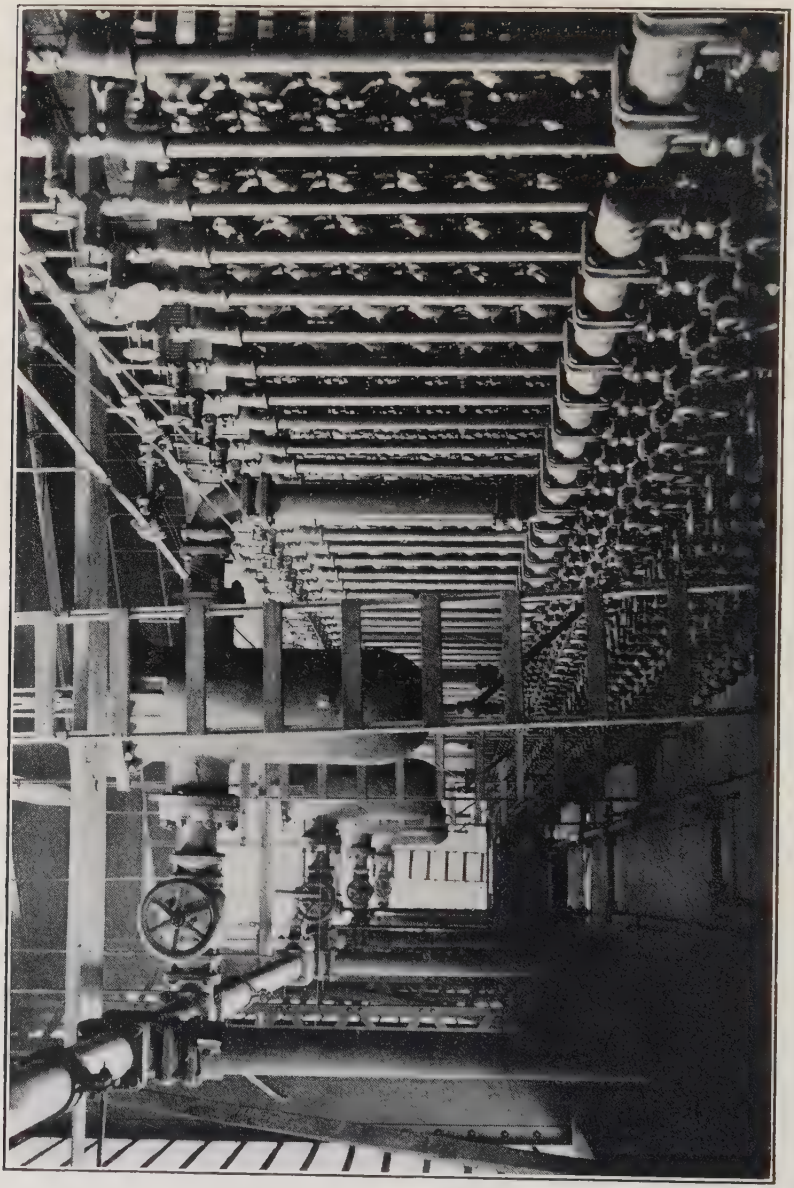


FIG. 2.—VIEW OF THE AMMONIA-CONDENSERS.

refrigerating-chamber, the brine is forced by one of two fly-wheel pumps. The cold brine enters the coils at the top.

The refrigerating-chamber contains 57 miles of 1¼-in. wrought-iron pipe, divided into seven sections by partition-walls and doors, so that one section can be washed free from frost without affecting the other sections or increasing the moisture in the treated air.

A large Sturtevant fan was installed to furnish air to the refrigerating-chamber, and maintain therein a pressure of about 1 ounce per sq. ft., in order to insure a proper distribution of the air rising through the coils, and also to overcome the friction in the pipes to the blowing-engines, so that there shall be no doubt as to the proper filling of the blowing-tubs. It also guards against the entry of untreated air through possible leaks in the pipes. We have found, however, that the plant can be run for a short time without this fan.

It was not to be expected that a plant of such magnitude could be erected and put in operation without the discovery of faults in construction and workmanship. The problem presented was very different from the simple manufacture of ice, or the refrigeration of air at rest in cold-storage rooms. Upon the assumption that 4.5 tons of air are required to make 1 ton of iron, the production of 720 tons of iron a day calls for the treatment every hour of 135 tons of air, moving at a rapid rate, and varying both in temperature and in contained moisture. We had our share of troubles at the beginning, but we detected no faults which we could not readily correct. For several months past no trouble has been experienced in operation, and the dry-air plant is earning the reputation of being the only "sure thing" about the furnace.

Dry Blast on No. 2 Furnace.

Our first application of the dry blast was made on our No. 2 furnace, which was 100 ft. high by 22-ft. bosh and 15-ft. crucible and stock-line. This furnace had been in blast for nearly three years, and had made more than 500,000 tons of pig-iron.

The campaign, as a whole, had been satisfactory in amount of product and rate of fuel-consumption. Down to June, 1907, with the exception of the three summer months of each year, the fuel-consumption had been kept low and the iron satisfac-

tory. But for almost a year before that date we had had high pressures and irregular settling of stock; and this we had attributed to poor distribution, due to the loss of the stock-line, which we had found in very bad shape when we had a chance to see it while changing our bell in October, 1906.

In June, 1907, the furnace ran on one-half wind for two weeks, while the dry-blast connections were being made to the engines. This was a difficult matter, on account of the Kennedy air-valves on the blowing-tubs.

Upon starting up full, in July, the furnace worked much worse. Large blank charges were tried without effect, and attempts were made to locate a scaffold. The working became still worse in August, and only the hope of some benefit from the dry blast led us to keep the furnace running. We could not make a large enough percentage of basic iron to fill our orders under the existing market-conditions. Frequent slips occurred, some of which filled the tuyeres with cinder and iron, and caused stoppage in order to open the tuyeres and cinder-notch.

On August 8, two compressors being ready, the dry blast was applied. On that day the moisture per cu. ft. of air in the blast was reduced from 7 grains to 5, the next day to 4, and the third day to 3.

The first effect was to brighten the tuyeres, so that they shone like stars. The slag increased greatly in temperature, and smoked so that one could not stay in the cast-house while it was running. The iron ran so hot that even gray iron "sparked" all the way down the runner.

The driving of the furnace increased, and the engines had to be slacked every day in the attempt to keep the rate of driving the same, while the burden was increased daily to keep the silicon in the iron down to 1 per cent.

The third day the furnace stuck tight, and it was impossible to get the full volume of air in at a pressure of 25 lb. This trouble yielded to large blanks, but only for a day or two. The moisture was gradually reduced to 2 grains per cu. ft., and held constant at that point. The quality of iron improved, but tonnage and fuel-economy did not, on account of the sticking fast several times a week for from six to ten hours at a time. Every one advised blowing out, but the orders on hand made that course inadvisable; and our President insisted that the dry blast

receive further trial before giving up the furnace. Mr. Collins, of Isabella, said in disgust that he knew how to put the dry blast on a "running" furnace, but this one was not "running." Mr. Scott filled her with coke twice, but it did no good. She worked black lumps at the tuyeres all the time, cleaned the bosh from top to bottom at intervals, and occasionally filled many of the tuyeres with slag and iron.

It seemed as if the dry blast, requiring an increase in the burden, had aggravated the trouble due to bad distribution and irregular interior-lines. In September the moisture was reduced to 1.5 grains per cu. ft. of air; in October, to 1 grain; and during November and December, to about 0.8 grain.

The tonnage of daily product remained low on account of frequent stops, and our efforts to keep the iron made within "basic" specifications. These specifications are: silicon under 1; sulphur under 0.05; and phosphorus under 1 per cent. But in order to have a good reputation, we try to keep these elements below 0.7, about 0.03, and below 0.8 per cent. respectively. As a whole the furnace was worn out. The top gave way twice, and all the gas went out of holes in the platform instead of down the down-comers. The stoves and boilers caused many stops. The fuel-consumption was increased by the numerous big coke blanks necessary to lower the pressure, so that the blowing-engines could run.

In spite of all these difficulties the furnace was kept running, and her work steadily improved until she was blown out on December 27, on account of our accumulation of basic iron, due to the closing of steel-works. There were short periods of good work (when the pressure stayed down, and there were no stops), in which 515 tons a day were made on 2,000 lb. of fuel per ton for several days in succession.

Figs. 3 and 4 show the condition of No. 2 when blown in and when blown out. In Fig. 4, the stock-line, 22 ft. 8 in. in diameter, is shown, with a large shelf below it, where the brick had stopped disintegrating on account of a change in quality. Further down there was another shelf, made up of scaffold-material. The scaffold did not fall until a week after the furnace had been cleaned out, and then filled her hearth 5 ft. above the tuyeres, so that she began to yield gas and had the appearance of being in blast again. We never imagined

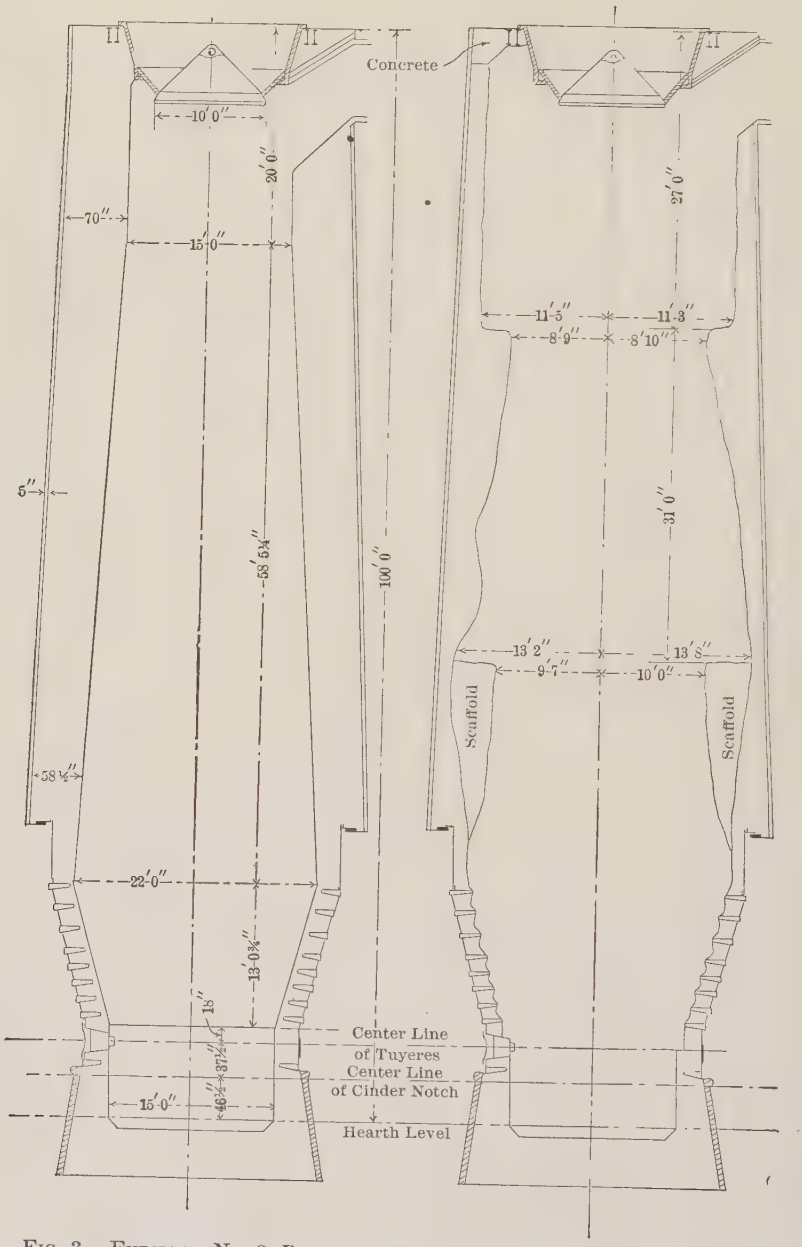


FIG. 3.—FURNACE NO. 2, BLOWN IN NOV. 1, 1904.

FIG. 4.—FURNACE NO. 2, BLOWN OUT DEC. 27, 1907.

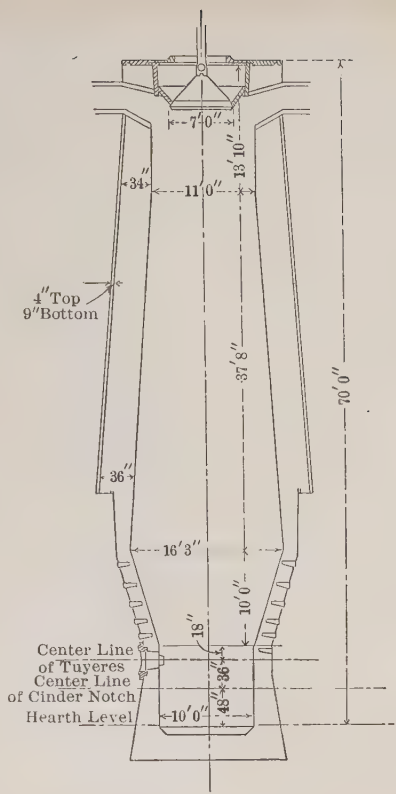


FIG. 5.—FURNACE NO. 1, BLOWN IN AUG. 3, 1905.

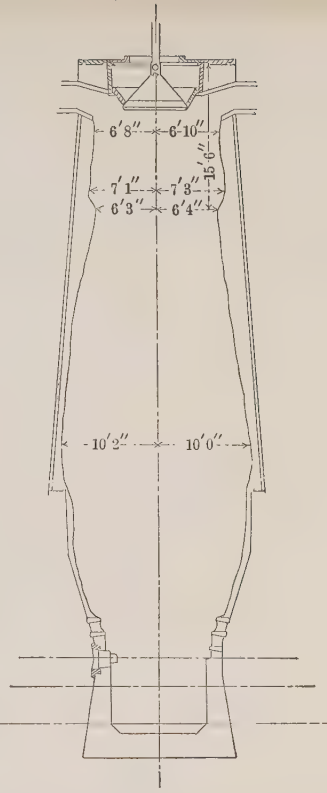


FIG. 6.—FURNACE NO. 1, BLOWN OUT NOV. 11, 1907.

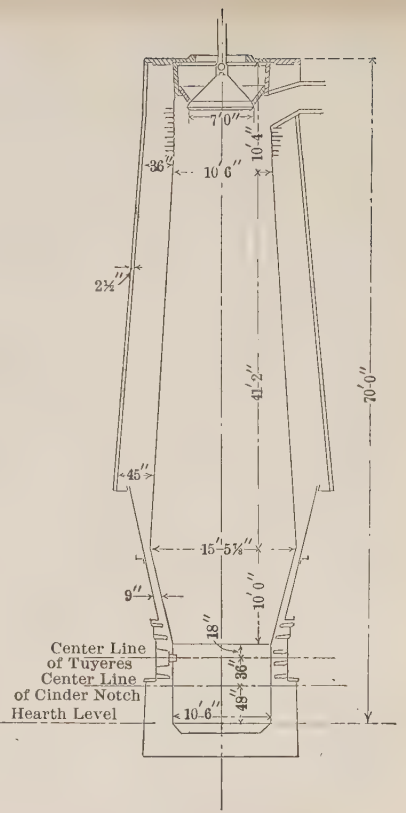


FIG. 7.—FURNACE NO. 1, RE-LINED, BLOWN IN MAR. 12, 1908.

that the scaffold could be so high up, and consequently did not find it, since we did no drilling higher than 4 ft. above the mantel. Three times during the summer the furnace had been blown down to below the mantel.

The following is the record for the last six months of the blast. The increased percentage of basic iron and the lowering of fuel-consumption, in a furnace without a stock-line, and with a large ring-scaffold, are remarkable.

Record of Furnace No. 2.

1907.	Fuel Per Ton of Iron. Pounds.	Quantity of Iron Per Week. Tons.	Proportion of Basic Iron. Per Cent.	Average Stoppages Per Week, in Minutes.
July, ...	2,534	3,025	89.2	194
Aug., ...	2,452	2,649	69.0	528
Sept., ...	2,503	2,638	78.2	441
Oct., ...	2,501	2,927	82.2	336
Nov., ...	2,339	2,735	80.4	336
Dec., ...	2,261	2,880	88.5	319

No. 2 furnace has since been repaired, but up to the present time (August, 1908), trade-conditions have not been such as to render it advisable to blow in. We have not, therefore, had the opportunity to test the virtues of the dry-air blast on the newly-lined furnace.

Dry Blast on No. 1 Furnace.

On September 25, four of the compressors having been completed, the dry blast was put on No. 1 furnace. In three days the moisture per cu. ft. of air was reduced from 6.5 grains to 1.50. There was the same evidence of greatly increased heat, shown by tuyeres, slag, and iron, as at No. 2 furnace.

No. 1 had been in blast for 2.5 years on high-silicon foundry-iron. The settling for the first year had been entirely by 4-ft. jumps, and the fuel-consumption had been consequently high, and the iron irregular in grade. After the first year, the three top rows of the total five rows of bosh-plates were pulled out of the furnace. This resulted in regular settling on foundry-iron and lower fuel-consumption.

In March, 1907, however, an attempt was made to turn this furnace to the production of basic iron. It proved impracticable, because an arch formed on the step left by pulling out the plates. The furnace stuck fast for 24 hr., and would not settle at all,

so that she had to be put back on foundry-iron. During the summer of 1907 her work became extremely bad, and the iron was irregular in grade.

Upon the application of dry blast in September, the furnace acted much as she had done in March, when the attempt was made to make basic iron. That is to say, the reduction of fuel (or, what is the same thing, the increase of burden) and the consequent lowering of the zone of fusion brought about a tendency to arch at the offset on the bosh.

The daily product increased about 10 per cent., and the quality of the iron improved somewhat. The fuel-consumption had to be kept relatively high in order to run the furnace at all. She was blown out on November 11, on account of our accumulation of foundry-iron. The dry blast had been used on her for 45 days, hardly long enough to get significant results from an old furnace. If the market had warranted keeping her in operation, there is little doubt that she would have improved. Fig. 5 shows her original, and Fig. 6 her wear-lines and the step on the bosh.

The effect of the dry blast on these two worn-out furnaces was a matter of great interest to the management. The first effect was to make them more troublesome, but this was explained by later experience as due to the too sudden application of the dry blast. The improvement in their work was slow by reason of the necessity of forming gradually somewhat different interior lines.

No. 2 was eventually properly burdened; but, in the light of recent experience, we are wondering whether No. 1 would not have done better had we had the courage to put on a heavy burden, in spite of the shelf on the bosh.

Dry Blast on No. 1 Furnace Re-lined.

No. 1 furnace was re-lined as shown in Fig. 7, there being practically no change in the lines, except that the crucible was made larger in diameter. The bosh is held by two rows of plates with a bosh-jacket above, on which there is 9 in. of brick. The idea is to avoid the cooling on the upper bosh (which seemed to cause the stock to settle irregularly during the previous blast), and still to be able to maintain the lines of the

bosh, so that basic iron can be made if desired. The furnace was lighted Mar. 12, 1908.

Some changes had been made in the ammonia-end of the dry-blast plant, and as the plant was not perfectly ready for use, the furnace was started on natural air, averaging, at the time, 4 grains of moisture per cu. ft. The blow-in was excellent. Ten days after the first iron was made, the dry blast was applied. The moisture was lowered from 4.5 grains, on Monday, March 24, to 1.5 grains on Tuesday. The revolutions of the engine were reduced 10 per cent. at the same time, and the burden was increased 5 per cent. The effect was immediate. At 9 a.m., when dry blast was applied, the silicon in the iron was 2 per cent. At 5 p.m. it was 2.75 per cent. Increased heat in the hearth was indicated by the appearance of tuyeres, slag, and iron.

The rate of driving decreased but little. That night the furnace stuck fast; the blast had to be thrown off to get her down, and she began to become cold. The moisture was then lowered to less than 1 grain per cu. foot.

The next day the sticking continued, and the pressure went up to 15 lb., at times stopping the engine. The use of cold blast did not lower the pressure.

It was necessary on these occasions to relieve the pressure entirely by throwing off the wind. The furnace would then settle; but the trouble recurred, sometimes at once, sometimes not until after 12 hours.

The iron got hard and the cinder black; but both ran very hot. The burden was reduced 7.5 per cent., and the blast was increased; but that made matters worse.

After five days of unsatisfactory work, we came to the conclusion that the furnace was underburdened. The sudden application of dry blast had greatly increased the heat of the hearth. This increased temperature, not being met by increased work in the form of heavier burden, necessarily worked high up in the furnace, causing sticking at the top of the bosh. The CO in the gas increased to 32 per cent. The burden was increased 15 per cent. in one change. When this came to work, the pressure held constant, and the furnace has been settling regularly since.

If there was any one thing that proved the power of dry

blast to us it was this experience in putting it on too fast. We had seen days when the moisture varied 3 grains per cu. ft. of air from the day before; and as we were in a hurry to get the benefit of uniform dry air, we thought it safe to try a sudden application.

Mr. Gayley and Mr. Scott advised us against sudden application, but did not give their reasons.

At Isabella they always take at least ten days, putting on one-quarter, and slowly burdening up, and so on. At Cardiff they followed Isabella practice. We have learned our lesson, and in the future will blow-in with dry blast if possible, and, if not, will put it on very slowly.

The next week, the fourth of the blast, the furnace made 1,311 tons, on 2,090 lb. of fuel. She has since, in several different weeks, made nearly 1,700 tons a week on less than 1,900 lb. of fuel, with heats under 900°, and an ore-mixture of 50 per cent. magnetic concentrates, and only 25 per cent. of Lake ore. She has made 1,400 tons of iron, averaging 2 per cent. of silicon, on less than 2,200 lb. of fuel per ton, on the same mixture, but with 1,000° F. of blast-temperature.

The monthly records are as follows:

Records of No. 1 Furnace.

1908.	Product Per Week. Tons.	Proportion of Foundry-Iron. Per Cent.	Yield of Ore. Per Ct.	Blast-temperature. Degrees F.	Fuel Per Ton of Iron. Pounds.
April, . . .	1,392	3	57.5	882	2,061
May, . . .	1,532	21	56.7	920	2,059
June, . . .	1,451	77	57.1	968	2,131
July, . . .	1,501	32	57.4	907	2,006

April was practically the first month of the blast, and the early troubles occurred in its beginning.

In May and June much high-silicon foundry-iron was made, some of it carrying more than 3 per cent. of silicon.

In July, there were a number of small stoppages, and one of more than 18 hours.

Warwick is a merchant-furnace, and the iron made must suit the order-book. At the present time the changes of grade are consequently frequent, and the convenience of the furnace is not considered.

We have no comparative records on basic iron, the nearest

being the record of forge-iron, in 1898, when the ores were entirely different, and the blast-temperatures much higher.

Product Per Week. Tons.	Yield of Ore. Per Cent.	Heat of Blast. Degrees F.	Fuel Per Ton of Iron. Pounds.
1,246	61.21	1,234	2,378

The best previous record of recent years on foundry-iron is that of 1906, as follows:

Product Per Week. Tons.	Yield of Ore. Per Cent.	Heat of Blast. Degrees F.	Fuel Per Ton of Iron. Pounds.
1,016	57.3	1,016	2,633

We feel that, at a moderate estimate, we are saving 400 lb. of fuel per ton of iron, and making 350 tons more iron a week, notwithstanding our lower blast-temperature, and a much more refractory ore-mixture. Although the fuel-economy shown is not as great as that exhibited by the Isabella furnace, comparatively it is as good. It is a remarkable fact that all the furnaces using dry blast are showing about the same percentage of saving of coke and increase of product as compared with their previous work.

The dry blast is by no means a cure-all. Although it greatly increases the regularity of the furnace, and enables it to be run closer, this very fact of doing away with the margin of safety, and running on a small slag-volume and a siliceous slag, makes the furnace more sensitive to changes of coke and variations in the character of the ore-mixture.

The settling of No. 1 furnace is very uniform, and she keeps level on top. On changing from one kind of iron to another we have irregularity, and occasionally have to slack her; otherwise she settles regularly. It is most unusual for her to make a jump, even during a cast.

The blast-pressure varies from 9 to 10 lb. per sq. in., according to variation in blast-heat, and the use of three different kinds of coke. The dry-blast plant is about 150 ft. from the No. 1 blowing-engine, and the pipe is not protected from the sun. The temperature of the air arriving at the engine varies about 15° in summer, between day and night. This makes a difference of, roughly, 5 per cent. in the volume of air that the blowing-engine delivers. The effect upon the furnace is to make her drive about 5 per cent. faster at night. On one or

two occasions this faster driving has lowered the silicon in foundry-iron; otherwise it has given no trouble. An attempt was made for a time to run the engine 5 per cent. faster on the day-shift if the sun was shining, or 2.5 per cent. faster if the day was cloudy. But this was soon given up, as it was too complicated, like the changing of blast-temperature to meet variations in humidity.

On one occasion the dry-blast plant was stopped from 8 a.m. to 6 p.m., to make a change in the brine-piping. The moisture rose from 0.63 to 4.08 grains per cu. ft. of air. The silicon in the iron made dropped from 2.5 down to 2 per cent., and then went back to 2.75 per cent., when the moisture was reduced again below 1 grain per cu. foot.

Sometimes the first flush of cinder after a cast was gray and the next black. This was invariably followed by a cast much higher in phosphorus, and somewhat higher in sulphur. There was no slip nor any indication of a slip on top, or in the gas, and we attributed the phenomenon to the occasional running ahead of the high-phosphorus magnetic concentrates. This trouble was partly corrected by a re-arrangement of the filling, and the making of a slightly more basic slag. A steam-reducing valve has caused many stops, and trouble with steam at the engine has reduced the tonnage for certain periods.

When the furnace is running on foundry-iron, the product is wonderfully uniform in silicon and sulphur, as between one cast and another, or different portions of the same cast. On basic iron we can safely carry our silicon from 0.2 to 0.5 without fear; and the sulphur can be held more constant than in a furnace blown with natural air. On two occasions, through a desire to lower fuel-consumption, the furnace has been overburdened, and hard iron and buckshot have resulted; but a small amount of extra coke and a reduction of burden to the proper amount have brought the furnace back on good iron immediately.

There are, of course, two ways of running a furnace: first, carefully, to make a regular grade of iron; second, up to the limit, to make large tonnage and reduce fuel-consumption, without reference to maintaining a given grade of iron.

Running in the first way with the dry blast, it is possible, with ordinary care in keeping the stock regular, to make every

cast of the grade desired. Running in the second way, some variation in silicon and sulphur occurs in the product of the dry blast, but not nearly so much as with natural air, irregular in moisture.

The gas stays high in CO (up to 26 per cent.) on account of the use of 50 per cent. of magnetic ore. The temperature of escaping gases on a 70-ft. furnace averages 300° F. on basic and 500° F. on foundry-iron.

On account of the regular settling, the flue-dust must necessarily be less; but our ore-mixture has been so changed that it is difficult to make comparisons. We are inclined to think that it was a mistake to make the stock-line so small in diameter, since, with the fine mixture now used, the amount of flue-dust made is considerable. It varies from time to time, even on the same mixture. For several days the amount made is excessive, as if the furnace were channeled; then it decreases again.

Detailed records have not been given, because I feel that they are of no value unless studied carefully in connection with a complete knowledge of the individual furnace and the daily circumstances.

Explanations.

Many furnace-managers, who have seen the Warwick dry-blast plant, have asked how we explain the results obtained. The following hypothesis satisfies us, and may be of interest to others:

Of all the heat-value of coke that goes into a furnace only about one-fourth is available for heating the hearth, about one-half is sensible heat in the gas, about one-quarter reduces the ore, some is required to carburize the iron. Now the hearth is the only limit, the only place where the ordinary furnace requires more heat. It was estimated for years, that dry blast would save only 3 per cent. of coke, but the fact was neglected that this 3 per cent. of the total fuel would be saved to the hearth alone, and that this would amount to four times as much, that is, 12 per cent. of the heat of the hearth, and as the hearth is the only weak place, the efficiency of the furnace would be increased 12 per cent. It is like a chain with one weak link—you increase the weight, say, 3 per cent. of the

total chain, but put all the increase of metal in the weak link, and increase the strength of the chain 12 per cent.

In this case, the dry blast decreases the duty of the hearth 12 per cent., and allows the resultant heat to be utilized upon increased quantities of iron and slag.

Further, this increased burden, together with regularity of heat supplied to hearth, enables the slag-volume to be decreased and the slag made more acid. This again saves fuel. The increased product decreases the radiation per ton of iron, and makes another saving. This will vary with individual furnaces, and will be particularly great on those that will stand more wind. I believe that a very close estimate can be made on the amount of saving dry blast will make on each furnace.

The temperature of a furnace is increased enormously when dry blast is applied, particularly if it is foolishly put on too fast, as on the No. 1 furnace, but the increased temperature is soon equalized with burden and more work, and the temperature kept at the desired point as in any furnace.

Costs.

Some estimate of the cost of operating the plant may be of interest. Based on a product of 720 tons a day, and running all the ammonia-compressors, the labor-cost is 2.5 cents a ton. In the fall, winter, and spring months it is 2 cents a ton. The oil, waste, etc., should not exceed a maximum of 1.5 cents a ton. Estimating on an excessive loss of ammonia (one-third of the entire contents of system yearly), the ammonia-cost will be 1 cent per ton. The brine-cost is insignificant. We estimated that 10 cents a ton would cover the cost of operation and maintenance; and we now feel that this will be more than ample.

Judging from the experience of general refrigerating-machinery and plants, an annual depreciation-charge of 5 per cent. will be sufficient.

No charge is made for steam; for, in our experience, the dry-blast plant has made no additional firing of coal necessary. The reduction in revolutions of the blowing-engines spares enough power to run the refrigerating-machinery, brine-pump, and fan. Although the gas is leaner, and of lower temperature, so that it is not as good fuel, its flow is regular; and there is no

sticking after casting, with loss of gas and consequent heavy firing.

SUMMARY.

The management of Warwick feels fully satisfied with its adoption of the dry blast. No explanation is needed to show its great value during this summer, of extremely high and irregular humidity, at a time when the iron market has been totally demoralized.

We are now installing new Roberts center combustion-chamber stoves at the No. 1 furnace, in order to obtain 1,300° to 1,400° F. of heat in the blast. It is a matter still open for discussion; but we feel that the dry blast will make possible the use of higher heats, and thereby indirectly lead to an additional saving in fuel.

In conclusion, attention is called to the fact that the efficiency of the dry blast has been demonstrated on a small furnace, and that the results warrant the statement that its application will be of particular value to small furnaces, as it gives them the efficiency, and somewhat of the production of iron, of the larger furnaces.

The claims of Mr. Gayley have been more than confirmed by the experience of every plant that has installed the dry blast, whether the furnace was large or small, and whatever the fuel and ore-mixture, and the grade of iron desired.

The Pearce Gold-Separation Process.

BY HAROLD V. PEARCE, LONDON, ENGLAND.

(Chattanooga Meeting, October, 1908.)

THE fire which occurred in the fall of 1906, at the works of the Boston & Colorado Smelting Co., Argo, Colo., destroyed entirely the gold- and silver-refinery of the plant, and in view of the development of modern practice in the refining of copper-mattes, etc., the management decided that it would be inadvisable to rebuild that part of the works for the continued use of the process which had been conducted therein for so many years as a metallurgical secret. This decision probably removes altogether from the sphere of practice the "Pearce" method for the separation of gold. At all events, it removes the busi-

ness reasons which have prevented hitherto the publication of the principles involved. At the suggestion of Dr. Raymond, Secretary of the Institute, my father, Richard Pearce, the former manager of these works, has requested me to prepare a brief description of this process for the benefit of our members. Since I severed my connection with the Boston & Colorado Smelting Co. some time ago, I am indebted for some practical details to the present superintendent, Charles Rees, whose friendly assistance I here gratefully acknowledge.

With regard to the history of the inception and development of this process by its inventor, Richard Pearce, I cannot do better than quote freely from his own notes and papers.

History of the Pearce Process.[1]

"In 1874 my entire time was directed to the improvements and additions to the plant at Black Hawk. The changes which I had made in the construction of the furnaces at Black Hawk proved so beneficial that I was requested to introduce the improvements at the branch works at Alma. These changes were carried out as speedily as possible, and proved of great advantage as compared with the old system.

"The separation and refining of the gold from the matte now engaged my attention, and since previous experience in the treatment of gold-bearing matte and other furnace-products was practically of no value, it became necessary for me to work out the process which appeared to be best adapted to the local conditions at Black Hawk.

"The following is a general sketch of the various stages which were reached in solving this interesting and important problem of gold-extraction:

"1. The concentration of the gold in the form of a rich copper-gold alloy, known as 'rich gold-bottoms.'

"2. The refining of the rich alloy in a suitable furnace and converting it into a hollow shell-like condition by granulation in water.

"3. The oxidation of the granulated gold-alloy, by which means the copper is oxidized into a condition permitting it to be ground in a suitable mill.

"4. The solution of the copper in suitable tanks by sulphuric acid.

"5. The melting and refining of the gold residue.

"The above is a brief outline of the scheme which was at first adopted. The chief difficulties which presented themselves in carrying out this scheme successfully, were, first, the heavy expense of procuring a supply of sulphuric acid, for at that time the nearest acid-manufactory was at St. Louis, and secondly, the copper sulphate, which was a by-product, had to be shipped East to a market, necessitating a heavy freight tax. On this account it was decided to erect a small plant in Boston for the acid treatment, so that the work at Black Hawk was limited to the preparation and refining of the alloy, and the oxidation and grinding of the granulated products.

"This scheme was successfully conducted in Boston for about a year. The rich oxidized alloy, which was forwarded to Boston, was carefully assayed for gold, and

[1] Quoted from the notes of Richard Pearce.

the Boston house was debited with the contents as indicated by Black Hawk assays. We were met with all sorts of difficulties in arriving at a fair adjustment of the assays. The material contained on an average about 600 oz. of gold per ton, representing a money-value in gold of about $12,000 per ton. There were, however, so many annoying features connected with the Boston branch in the settlement of the assays, that I was driven in desperation to devise some method by which the whole business could be done at Black Hawk, and after a great deal of thought, an idea suggested itself to my mind which, if it could be practically carried out, would solve the whole difficulty.

"I started a series of experiments unknown to Professor Hill, to prove whether or not my scheme had any merit, and whether it could be worked on a practical scale. I soon satisfied myself on this point by producing a lump of gold weighing 2 or 3 lb. I took this specimen of my work to Professor Hill's office, and on handing it to him, his first inquiry was: 'Where did this come from?' I then explained to him how it was produced, and his next question was: 'Can you produce it on a large scale?' My reply being: 'Yes, and I will promise to ship a bar of 1,000 oz. next week.'

"Professor Hill, without further questioning, telegraphed at once to Boston instructions to work up whatever stock of rich material they had on hand, and close the establishment, for in future the refining would be carried out at Black Hawk.

"The plant erected in Boston for the separation of the gold from copper was subsequently utilized for the manufacture of copper sulphate from our Black Hawk residue, and for some time the bulk of the copper was marketed as bluestone.

"This solved the whole problem, for each metal was put into marketable form; the gold in bars ranging upward from 900 fine and sold to the U. S. Mint; silver in fine bars 999 fine, sold to the Mint, this silver having the reputation of being the finest then produced in the United States; copper as marketable copper sulphate.

"The success of these methods enabled the company to pay increased prices for all ores which contained little or no lead, so that the business rapidly increased in volume.

"It is just 30 years ago since this process was worked out, and it is still in use at this time (1905). I find, on referring to my note-book, that from the date of its commencement to the closing down, on account of the fire, the extraction of gold by the 'Pearce' process amounted to 1,532,905 oz. (52.55 tons), obtained from 54,549 tons of copper."

Before attempting to describe the process in detail it is necessary to outline the sequence of operations preceding the application of the Pearce process for the final separation of the gold from a copper-gold alloy; therefore I introduce here quotations from the Presidential address to the Institute of Richard Pearce at the Colorado Meeting, 1889, in which are fully described the general operations as conducted at Argo. In this address the description ended at the point where the secret process began, owing to the objection of the management, for business reasons, to publish a detailed description of the final separation. For convenience, I quote only from that portion of my father's paper which has special reference to the stage in the operations which bears upon the

present paper, beginning at a point where the treatment of residues is described after the major portion of silver has been removed from the copper-matte by the "Ziervogel" process.

Treatment of the Residues.[2]

"The residues, after the extraction of the silver, contain about 40 oz. of silver and 10 oz. of gold per ton, and 55 per cent. of copper in the form of cupric oxide. They also contain associated metals, such as lead and bismuth as sulphates, and iron as ferric oxide.

"The residues are mixed with the necessary quantity of pyritous ores containing gold, or low grade silver-bearing pyrites rich in sulphur, and with siliceous gold-ores, and are smelted direct for matte of 65 per cent. copper. This matte contains about 10 to 15 oz. of gold per ton and 80 oz. of silver. The slag from this operation is thrown away. It contains about the same proportion of iron and silica as the ore-slag.

Treatment of Matte for Rich Copper-Gold Alloy.

"This process involves two operations, which may be described under the following heads: A combined roasting and smelting, and a refining of crude alloy.

"The aim of this process is to concentrate the gold contained in the matte into a rich gold-silver-copper alloy, containing also the bulk of the impurities existing in the matte. This process resembles the old Swansea process for making 'best selected copper,' a description of which will be found in any of the standard works on copper-smelting. As practiced at Argo, it is as follows:

"Twelve tons of matte, or residue metal, are placed in slabs on the hearth of a large reverberatory furnace. The openings of the furnace are closed, except two large ports, which are situated near the bridge, and are used for the supply of air for oxidation. The heat is gradually increased for a period of about seven hours, or until the whole mass assumes a semi-pasty condition. The ports are then closed, and the heat is still further increased until a point is reached, when the whole charge is in a perfectly molten condition. In the last stage of the melting a reaction takes place between the oxide and sulphide, and a certain amount of metallic copper is liberated. This metallic copper contains nearly all the gold, together with any foreign metals present in the matte, such as lead, arsenic, antimony, bismuth, etc. The charge is tapped into sand molds, in the first few of which, nearest the taphole, there will be found, after cooling, under the enriched matte, plates of the above-mentioned impure copper, commonly known as metallic bottoms. The proportion of these bottoms to the whole tap of matte, called pimple metal, is about 1 to 15. The following analysis will give some idea of the composition of this crude alloy:

	Per Cent.
Copper,	60.04
Lead,	33.61
Arsenic,	0.44
Bismuth,	0.40
Iron,	0.08
Zinc,	0.15
Gold,	0.54
Silver,	1.35
Sulphur,	1.68
	98.29

[2] Extract from Presidential address of Richard Pearce, *Trans.*, xviii., 68, 69 (1889–90).

"It must be understood that the analysis shows the composition of one specimen of such alloy only. The composition varies, of course, with the nature and quantity of the impurities present in the residue metal. These impurities, without a doubt, increase the solvent action of the copper on the gold. In other words, a copper matte containing little or none of the impurities mentioned, will not yield its gold so readily to the copper. The whole operation of roasting and smelting occupies about twelve hours. The crude bottoms contain 100 to 200 oz. of gold to the ton, and about 300 oz. of silver. The compound that bismuth forms with gold, when present in this crude alloy, has been referred to in my paper on 'Certain Interesting Crystalline Alloys.'[3] The matte, pimple metal, if the operation has been properly done, contains not more than 0.2 oz. of gold per ton (assays frequently show less than 0.1 oz. of gold), 90 oz. of silver, and 77 per cent. of copper. The roaster slag from this portion is smelted in the ore furnaces.

Treatment of the Crude Alloy.

"The treatment of this alloy and the separation of the gold from the copper is a part of the process which, for certain business considerations, I am not at liberty to describe."

Description of the Pearce Process.

If the gold-copper alloy in the form of copper-bottoms be granulated and melted with a suitable quantity of pyrite (FeS_2), there results a matte and a copper-bottom. The matte is the result of the combination of the iron sulphide with a portion of the copper of the alloy, and the bottom is the remaining portion, carrying the bulk of the gold present in the original alloy.

This bottom, being melted, refined, and again granulated, is once more melted with pyrite, a further quantity of copper combining with it to form matte, and a smaller amount remaining as metallic copper, still carrying the major portion of the gold.

It will now be readily seen that the treatment of any copper-gold alloy involves a series of refinings with granulations and subsequent treatments or "strippings" with pyrite, with the result that the whole of the copper is finally extracted or "stripped" from the gold. A certain quantity of gold passes into the matte at each stripping, but any attempt to remove too large a proportion of the copper in one operation results in too much gold being taken up by the matte.

The operation is conducted in the following manner:

The copper-gold alloy in the form of bottoms, assaying from 125 to 150 oz. of gold per ton (0.4 to 0.5 per cent.), is piled up

[3] *Trans.*, xiii., 738 (1884–85).

on the brick hearth of a small reverberatory furnace, in charges of about 3 tons, and melted down very slowly, usually in about 8 hr. During this time air is introduced at the side doors and through the port-holes situated at either side of the furnace near the fire-box, and the sweating and melting-down of the alloy is accompanied by the oxidation of the lead and other impurities, together with a certain quantity of copper. Toward the end of the operation the temperature is increased, and a large quantity of slag is formed, consisting largely of lead silicate, with CuO and Cu_2O. The CuO acts as a carrier of oxygen to the impurities present, such as arsenic, antimony, and sulphur, which still remain under the surface of the slag, the CuO being converted into Cu_2O. At the end of this operation the ports and side doors are closed, and the charge raised to a higher temperature until thoroughly melted and quite clear from the bottom. The slag is then carefully skimmed into ordinary sand-molds, and the alloy is tapped into water and granulated.

As stated before, the treatment of this granulated product is the key of the process, and depends on the action of iron sulphide on copper when melted together, whereby a definite compound of copper with iron and sulphur is formed, corresponding closely to the mineral bornite both in appearance and composition. This matte is locally called "red metal."

The stripping is accomplished in another small reverberatory furnace, in charges of somewhat less than 2 tons of mixed alloy and iron sulphide, the latter, usually in the form of clean pyritic concentrates, containing about 10 per cent. of SiO_2. It may be mentioned that pyrrhotite does not answer as a substitute for pyrite.

The usual practice is to add sufficient iron sulphide to reduce the 2,200 lb. of granulated copper to about 650 lb. of bottom, the charge generally consisting of 2,200 lb. of copper and 1,500 lb. of pyritic material.

After melting and tapping in sand-molds, the metallic bottoms are found under the first few pigs of matte nearest the tap-hole. These bottoms are once more granulated, stripped, and the still further concentrated alloy, in the form of bottoms, is removed to the refinery department proper, where the operations are carried out in plumbago crucibles in air-furnaces of

the usual type, the richness of the alloy varying from 18 to 30 per cent. of gold. In this department a precisely similar series of operations is carried on, only with smaller quantities. The alloy usually requires three strippings and three refinings to yield a finished bar of a fineness of 900 or more. Occasionally, however, the alloy comes into the refinery so rich in gold that it can be finished in two strippings and two refinings, the final refining of gold taking place in the usual manner of gold-refining in crucibles with niter, the product being poured into suitable bars for disposal to the U. S. Mint.

Description of Practical Working of Process.

Treatment of Copper-Bottoms from Roaster Furnace.

A fair charge of 6,400 lb. of copper-bottoms is placed on the brick bottom as near the bridge-wall as possible. Firing is commenced with port-holes open, the copper gradually melting and dripping down. When the charge is all melted, the port-holes are closed and the fire is forced. When the proper pitch of the copper is reached, the slag is skimmed off, after which the copper is tapped through a cast-iron launder at the side of the furnace into a tank or pit of cold water, the stream of copper falling on to a spruce pole, which sprays it before it strikes the water. The granules are caught on a perforated pan at the bottom of the pit, which is then hoisted, the water escaping through the perforations. A charge of this first granulated alloy is weighed and mixed with iron pyrite as follows: alloy, 2,200, and pyrite, 1,500 lb. The quantity of alloy from one granulation makes about two charges of mixture for stripping.

After a thorough mixing, the 3,700 lb. of granules and pyrite is charged into an adjoining furnace for the first stripping. When melted, the slag is skimmed off, and the matte (a red metal) is tapped into a series of sand-molds at the side of the furnace, and in the first three molds under the matte the copper-bottoms are found, weighing altogether from 625 to 650 lb. The first-granulation process is repeated until from 10 to 14 charges have been worked through the granulating-furnace, yielding for the first stripping a total of from 22 to 30 charges.

The copper-bottoms from the first stripping are now charged into the granulating-furnace and worked up in like manner for

the second granulation—the size of the charges being regulated by the quantity of bottoms to be worked up. The operation is similar to that of the first granulation. A sample is taken from the alloy as tapped, which varies in richness from 2 to 3 per cent. of gold. The mixture for the second stripping is based on the richness of the alloy, more or less pyrite being added according as the alloy is poorer or richer. The weight of the alloy is always the same—namely, 2,200 lb. The essential conditions for successful operation are: a fairly clean slag, matte as low as possible in gold, and bottoms as rich as possible in gold. The weight of the pyrite in the charge varies from 1,300 to 1,400 lb. The following shows the desirable weight of bottoms in the second stripping from a given alloy:

Alloy. Per Cent. of Gold.	Weight of Bottoms. Pounds.
2,	440
2.9,	490
3.4,	510
3.7,	520

The bottoms obtained from the second stripping are charged into the granulating-furnace for the third granulation, or "finishing-charge," as it is called.

Great care is necessary with this charge. The furnace having sufficiently cooled after the second granulation, the bottom is carefully examined by the furnace-man, and all holes in the sides plugged with stiff fire-clay. The tap-hole is cut down to an exact level with the bottom and closed by inserting a plug of stiff clay. The furnace is now charged with the entire quantity of second-stripping bottoms, and firing is begun with open port-holes and partly open side-door; the charge is allowed to roast down very slowly, usually taking about 12 hr. As soon as the charge is nearly all melted, the firing is forced with port-holes and side-door still slightly open, and when it is thoroughly melted and free from bottom, a small tapping-bar with a sharp point is driven through the tap-hole at a point on a level with the bottom (the bar must be provided with a notched head so that it can be driven out again). When the tapping-bar is driven out, the rich gold-copper alloy runs out in a small stream down the clay-lined launder into the granulating-tank, without being sprayed by pole. The tap-hole must be closely

watched, and a few clay plugs on sticks kept in readiness to stop the tap-hole at the first appearance of the slag which follows the copper alloy. This slag is afterwards tapped into sand-molds. The pan is then raised and the alloy sacked, sampled, and carefully weighed before transferring it to the refinery.

Treatment of the Rich Alloy in the Refinery.

The operations in the refinery are as follows:

First. { Stripping. / Refining and granulating.

Second. { Stripping. / Refining and granulating.

Third. { Stripping. / Refining and pouring into bars ready for shipment.

These strippings and refinings are carried out in plumbago crucibles in an ordinary air-furnace. In these operations the pyrite used is the cleanest variety possible to obtain.

The stripping-charge, consisting of 40 lb. of granulated alloy, and from 16 to 20 lb. of FeS_2 (the amount varying according to the richness of the alloy and the amount of FeS_2 in the material), is first melted in crucibles with sufficient borax to make a fluid slag, the content of each crucible is then poured into a conical cast-iron mold and allowed to cool. On inverting the mold the enriched copper-bottom is found at the apex of the cone, the matte and a small amount of slag forming the remainder. This copper-bottom is easily broken away from the matte, and is in convenient form for the next refining. This refining is done in the ordinary manner with niter, and the granulation by dipping out the alloy with a small ladle and pouring into a barrel of cold water placed conveniently near the furnace.

The skimmings during refining usually contain a large quantity of bismuth, and these are collected during treatment in one lot, and are subsequently melted in a crucible, to yield a bottom consisting mostly of bismuth with other impurities, and containing a varying percentage of gold. Bottoms of this kind weigh from 100 to 200 oz., and carry from 2 to 10 per cent. of gold. Several strippings are necessary, since if too large a proportion of the copper be removed at one time by the addi-

tion of a larger quantity of FeS_2 the resultant matte will carry with it too much gold.

Even when three comparatively light strippings are employed, the matte takes up a considerable quantity of gold, as will be shown by the data given later in this paper.

To illustrate the process in its final stage, the following is an example of a lot of rich alloy, as put through at Argo:

Weight of charge of alloy,	1,538 lb.
Gold-content,	12.45 per cent.
Total gold-content,	191.481 lb. (2,791.77 oz.)
First Stripping:	
Charge,	alloy, 40 lb.; FeS_2, 19 lb.
Weight of each bottom,	17½ lb.
Total weight of bottoms,	670 lb.
Total weight of matte,	1,575 lb.
First Refining:	
Product,	granulated alloy, 612 lb.
Gold-content,	25 per cent.
Second Stripping:	
Charge,	alloy, 40 lb.; FeS_2, 17 lb.
Weight of each bottom,	20½ lb.
Total weight of bottoms,	313 lb.
Total weight of matte,	550 lb.
Second Refining:	
Product,	granulated alloy, 300 lb.
Gold-content,	45 per cent.
Third Stripping:	

Since it often happened that it was necessary to proceed with this third stripping before an assay of the alloy could be obtained, it was customary to figure that 90 per cent. of the gold in the original quantity of alloy was present in this product of the second refining; 172 lb. of gold was therefore reckoned to be contained in this 300 lb. of alloy, the remainder, or 128 lb., being assumed to be all copper. Usually, 1 lb. of the FeS_2 material could be counted on to remove 1 lb. of Cu in the form of matte, the amount, of course, varying from time to time with the composition of the material employed.

The 300 lb. of alloy was therefore divided into:

Charges,	alloy, 50 lb.; FeS_2, 21⅓ lb.
Weight of each bottom,	417 oz.
Total weight of bottoms,	2,502 oz.

The rich matte from this stripping was cleaned by remelting with the addition of metallic iron, which displaced a certain

quantity of copper that carried down with it a large percentage of the gold. In this case three charges, each of 80 lb., were melted down with about 4 lb. of iron, a bottom of 373 oz. resulting, of practically the same richness as the third-stripping bottoms.

Third Refining:

Total quantity of alloy refined, 2,875 oz., containing fine gold,	2,791.77 oz.
Product, 2,796.7 oz. 920 fine, or fine gold,	2,572.96 oz.
Percentage of gold extracted,	92.1 per cent.

The following data, pertaining to three lots of crude copper-bottoms treated by the Pearce process, show the variation in richness of material treated from time to time:

Lot (x). First Granulation.

	Gold. Oz. Per Ton.		Gold. Oz. Per Ton.		Gold. Oz. Per Ton.
Tap 1,	145.83	Tap 6,	119.58	Tap 11,	104.99
2,	151.66	7,	118.99	12,	121.91
3,	102.66	8,	157.49	13,	106.16
4,	109.37	9,	106.64	Average,	121.02
5,	120.16	10,	107.91		

After the first stripping and second granulation the alloy averaged 2.1 per cent. of gold, or 612.5 oz. per ton.

The third granulation, or finishing-charge, produced 1,418 lb. of alloy, assaying 19.5 per cent. of gold, or 5,687 oz. per ton, and containing 4,031.51 oz. of gold to be treated in the refinery. The result was as follows:

Gold bars, 921 fine, shipped to the U. S. Mint:

	Ounces.
No. 1,429,	1,072.92
No. 1,430,	1,060.64
No. 1,431,	1,027.50
No. 1,432,	1,016.89
	4,177.95

The total gold is accounted for as follows: The total amount of the gold in the alloy was 4,031.51 oz., and the products were:

Gold in bars,	3,843.11 oz. =	95.30 per cent.
Gold in 1.235 tons of matte @ 76.9 oz. of gold per ton,	94.97 oz. =	2.35 per cent.
Gold in 135 oz. of bismuth-alloy @ 7.56 per cent. of gold,	10.20 oz. =	0.25 per cent.
Total,	3,948.28 oz. =	97.90 per cent.

Lot (y).

	Gold. Oz. Per Ton.		Gold. Oz. Per Ton.		Gold. Oz. Per Ton.
Tap 1, ...	198.32	Tap 5, ...	148.74	Tap 9, ...	139.99
2, ...	192.49	6, ...	134.74	10, ...	124.83
3, ...	152.25	7, ...	134.74	11, ...	127.74
4, ...	152.24	8, ...	151.07	Average, . . .	150.65

Second granulation assayed 2.53 per cent. of gold.

Third granulation produced 1,051 lb. of alloy, assaying 13.63 per cent. of gold, containing 2,088.58 oz. of gold, to be treated in refinery. Results:

Gold bars, 935 fine, shipped, No. 1,471, . . .	1,151.1 oz.
No. 1,472, . . .	1,015.62 oz.
	2,166.72 oz.

Total gold accounted for as follows: Gold in alloy, 2,088.58 oz.

Gold in bars, ...	2,028.609 oz. =	97.00 per cent.
Gold in 0.975 ton of matte @ 101.01 oz. per ton, . . .	98.484 oz. =	4.71 per cent.
Gold in bismuth-alloy, 201 oz. @ 2.56 per cent. of gold, . .	5.145 oz. =	0.24 per cent.
Total, ...	2,132.238 oz. =	101.95 per cent.

Lot (z).

	Gold. Oz. Per Ton.		Gold. Oz. Per Ton.		Gold. Oz. Per Ton.
Tap 1, ...	586.23	Tap 5, ...	175.99	Tap 9, ...	192.37
2, ...	280.57	6, ...	207.07	10, ...	499.28
3, ...	221.07	7, ...	198.32	11, ...	166.82
4, ...	201.82	8, ...	187.24	Average, . . .	265.22

Second-granulation alloy averaged 3.5 per cent. of gold.

Third granulation and finishing: weight, 1,088 lb., assaying 18.17 per cent. of gold, containing 2,882.17 oz. of gold.

Result of treatment in refinery:

Gold bars, 925 fine, No. 1,398, ...	1,038.10 oz.
No. 1,399, ...	1,029.65 oz.
No. 1,400, ...	990.65 oz.
	3,058.40 oz.

Total gold accounted for as follows: Gold in alloy, 2,882.17 oz.

Gold in bars, ...	2,824.43 oz. =	97.97 per cent.
Gold in 0.934 ton of matte @ 86.5 oz. per ton, ...	80.84 oz. =	2.80 per cent.
Gold in 233 oz. of bismuth-alloy @ 2.83 per cent. of gold, . .	6.59 oz. =	0.23 per cent.
Total, ...	2,911.86 oz. =	101.00 per cent.

The following data show the percentage of extraction, and the division of the balance between the matte and the bismuth-alloy, in 125 other lots. The variation in composition is due to the impracticability of obtaining accurate assays from products so very rich.

Finished Alloy.

Number of Lots Averaged.	Gold in Alloy.	Gold in Bars.	Gold in Matte.	Gold in Bismuth-Alloy.	
	Per Cent.	Per Cent.	Per Cent.		
15	13.86	95.88	4.08	0.36	
10	16.378	96.81	2.92	0.26	At this period
10	17.46	96.54	3.15	0.29	termed "bis-
10	18.60	96.91	2.61	0.47	muth-alloy."
10	17.93	96.36	3.11	0.52	
10	18.69	94.66	3.81	1.03	
10	20.72	95.53	3.45	2.50	
10	18.83	97.88	3.52	3.46	At this period
10	20.65	94.93	2.80	2.50	termed "in
10	19.54	93.56	2.56	4.10	balance."
10	17.38	94.37	3.41	2.40	
10	13.84	95.39	4.30	2.02	
125 Aver.	17.823	95.73	3.31	1.65	

The comparatively small amount of bismuth-alloy is returned for re-treatment, the gold being eventually recovered therefrom and the bismuth eliminated. The matte carrying some gold is also re-treated and the gold extracted. The process was never materially changed from the time it was first placed in operation in Black Hawk until the works were destroyed by fire. It is quite possible that the simplicity of the method of separation accounts for the fact that it was not developed before.

The Distribution of the Elements in Igneous Rocks.

BY HENRY S. WASHINGTON, NEW YORK, N. Y.

(Chattanooga Meeting, October, 1908.)

I. Introduction.

During the last twenty years or so the chemical investigation of rocks has made great advances, and it is now generally recognized that a knowledge of the chemical composition is as essential as that of the texture or mineral composition—if not more so—for the proper classification of rocks and study of their origin and relationships. Rock-analyses have vastly increased in number and, what is of greater importance, in quality. New and improved methods permit of greater accuracy than was possible in the early days, and the list of chemical constituents frequently determined has risen from the seven or eight of the greater part of the nineteenth century, to twenty or more. Indeed, rock-analyses with determinations of so many constituents are now commonly made by the chemists of the United States and Australia, while in Germany, Great Britain, France, and Italy the rarer constituents are determined more frequently than formerly.

As a consequence of this modern, accurate work, it has been discovered that some elements which were formerly supposed to be rare are of wide-spread occurrence and are often present in considerable amount. The fact is further being developed that the elements tend to show certain relations of occurrence or abundance in connection with each other. This is a fact which is applicable to the rarer elements, and which also finds a broad geological and petrological expression in the recognition of petrographic provinces. We are beginning to obtain some definite, though as yet rudimentary, knowledge of the distribution of the elements among igneous rocks.

Some of the results along these lines obtained by study of the vast accumulation of analytical data now available are well known to petrologists, while others do not seem to be so gener-

ally understood. To the non-petrologist they are, naturally, mostly unknown, and, as the general principles involved, and, indeed, some of the specific instances, have a more or less important bearing on the occurrence and characters of certain deposits of metallic ores and other economically important minerals, a discussion of the subject may be of interest to mining engineers.

Indeed, to the observations and operations of mining engineers and mining interests generally, the petrologist is indebted and must look for some of his data. This is especially true of those relating to the precious metals and others of commercial importance, the amounts of which usually present in rocks are so small as almost or quite to defy detection by ordinary analytical methods, and whose presence is often revealed only through search for and the exploitation of localities where they have undergone concentration. It must be premised, however, that our knowledge is at present very uneven, allowing fairly safe and detailed generalizations as regards some of the elements, very rudimentary or general ones as regards others, and again allowing almost none at all.

II. General Chemical Composition of Igneous Rocks.

The first and most important fact to be noted of igneous rocks is that, with the exception of some rare ore-bodies due to the differentiation of igneous magmas, they are composed almost wholly of silica and silicates. The vast majority of igneous rocks are silicate rocks, in which silica forms the most prominent and the never-failing constituent. Most of the minerals which compose them are combinations of silica with various bases, and it is a striking fact that the number of minerals which go to make up the majority of igneous rocks, and which are most abundant and most often met with, is very small.

The proportions in which these minerals may be present vary very widely. Some rocks are known which are composed wholly, or practically so, of but one mineral. Combinations of two are not infrequent, while most rocks contain at least three, and usually many more, minerals and in the most widely diverse proportions. It follows, therefore, that the chemical composition of igneous rocks may vary within very

wide limits, as regards any or all of the chemical constituents; and that, furthermore, some rocks may be of very simple chemical composition, while others may be very complex, with many constituents present, since the minerals themselves may be either very simple or highly complex in chemical composition.

The most important chemical constituents (stated as oxides, in accordance with the usual custom) are as follows: silica, SiO_2; alumina, Al_2O_3; ferric oxide, Fe_2O_3; ferrous oxide, FeO; magnesia, MgO; lime, CaO; soda, Na_2O; potash, K_2O; and water, H_2O. Some or all of these major constituents, as they are termed, are invariably present, so far as known, and collectively they constitute about 98 per cent. of all known rocks. The chief oxides, from silica to potash inclusive, enter into the composition of the most important and most commonly occurring rock-forming minerals, as well as the glass of imperfectly crystallized rocks.

The rôle of water is somewhat different. It would seem to be universally present in the magma, and its presence (along with that of other substances) lowers the freezing-point and increases the tendency to crystallization of the liquid mass. Most of this water is lost if the magma reaches the surface and it appears in the enormous clouds which accompany volcanic eruptions and the steam of volcanic fumaroles; and much of it also escapes if the magma solidifies beneath the surface, giving rise to subterranean water-supplies, which are held by many to be an important factor in the formation of many ore-deposits. A small proportion of the water originally present may remain in the solidified rock in a combined form, as part of the more complex mineral molecules, those of pyroxenes, amphiboles, and micas, for instance; and some may also remain as inclusions of water in the minerals of intrusive rocks.

There are also almost invariably present in igneous rocks small amounts of titanium, phosphorus, and manganese, though these are often neglected and thus overlooked in the less complete analyses. Carbon dioxide is also met with, but its presence, as reported in analyses of igneous rocks, is almost invariably due to decomposition, and it cannot be usually regarded as an essential or original constituent of rocks.

In addition to these most important constituents, the refine-

ments and increasing completeness of modern rock-analysis show that many others are frequently present, often in scarcely more than traces, but again in very appreciable quantities. The most important of these minor elements are: zirconium, sulphur (as sulphides and as sulphur trioxide), chlorine, fluorine, vanadium, chromium, nickel, barium, strontium, and lithium. Exceptionally, others may be determined: as boron, cobalt, copper, gold, silver, molybdenum, the metals of the cerium and yttrium groups, nitrogen, and others. Indeed, as Dr. Hillebrand[1] says, "a sufficiently careful examination of these rocks would show them to contain all or nearly all the known elements, not necessarily all in a given rock, but many more than anyone has yet found." The researches of Sandberger, Stelzner, and Dieulafait also point to the same conclusion.

Considering the quantitative distribution of the major constituents, silica is almost invariably present in igneous rocks, and almost always in greatest amount. In general, the percentage may vary from nearly 80, as in granites and rhyolites, to a minimum of about 24 per cent. Indeed, it may even form 100 per cent., if some dikes consisting wholly of quartz are really of igneous origin, as is believed to be the case; while in some ores derived from igneous magmas the amount of silica may drop almost to zero. Alumina is usually the next most abundant constituent, the percentage varying from a maximum of about 60, as in some corundum-syenites of Siberia and Canada, to a minimum of zero, in certain rocks composed wholly of olivine. The two iron oxides each show maxima of about 15 per cent., except in the rare iron-ores due to magmatic differentiation, where they constitute together almost all the rock. Magnesia attains maxima of from 45 to 50 per cent. in dunites of New Zealand and North Carolina; while lime reaches a maximum of about 20 per cent. in the anorthosites of Canada. Iron, magnesia, and lime may be practically absent in highly siliceous rocks, like granites and rhyolites, and in some syenites. Soda may be present up to 17 per cent. in some rare rocks in which nephelite is abundant; while potash attains a maximum of only about 12 per cent. in some unusual rocks rich in leucite,

[1] *Bulletin No.* 305, *U. S. Geological Survey*, p. 20 (1907).

which are found in Italy and in Wyoming. Both the alkalies may be wholly wanting in rocks composed essentially of pyroxene or olivine.

The amount of water present in wholly crystalline rocks seldom exceeds 2 per cent., if the rock is unaltered, though weathering very materially increases this quantity, and high figures for water are usually to be attributed to this cause. But some perfectly fresh, glassy lavas may carry up to 12 per cent. of water, which was unable to escape from the magma owing to the rapidity of solidification.

Titanium dioxide may rarely reach figures of about 6 per cent., as in some basalts of the Western Mediterranean which I am now investigating, but is usually present in much less quantity, though it is seldom or never entirely absent. In some titaniferous iron-ores of igneous origin, as those of the Adirondacks and Norway, it may even reach 18 per cent. The amount of phosphoric pentoxide rarely exceeds 3 per cent., and that of manganous oxide is scarcely ever more than 1 per cent., the higher figures sometimes reported for this latter constituent being almost invariably due to errors of analysis. It is but seldom that either of these three elements is entirely absent.

The maximum amounts of the other minor constituents may be briefly stated, as attention will thus be called to their relatively great rarity, it being understood that the maxima are seldom attained and that very frequently these elements are wholly absent.

Zirconium is much less common than the chemically related titanium, and seldom exceeds 0.20 per cent., though in some very exceptional cases it may reach 2 per cent. Chromium seldom occurs in amounts above 0.5 per cent., though a few rocks are known in which it is reported to range between 2 and 3 per cent. Nickel seldom exceeds 0.20 per cent., and the allied cobalt is scarcely ever present in more than mere traces. The maximum amount of copper found in unaltered igneous rocks may be placed at about 0.2 per cent. of CuO. Barium almost always exceeds strontium in quantity, but only very exceptionally exceeds 0.25 per cent., though some rocks are known in which about 1 per cent. is present; while the amount of strontium is usually much less than 0.1, but may occasionally reach 0.3 per cent. in the rocks very high in barium.

Although figures of 0.1 or 0.2 per cent. have been reported for lithium, these are somewhat doubtful, and it seldom occurs in more than spectroscopic traces. Sulphur and chlorine may both be present up to 2 per cent. or slightly more, but both usually occur only in tenths of a per cent., and the latter amount is the maximum for fluorine. Of the other minor constituents the amounts present are so small as usually to be insignificant except as to their actual presence.

III. The Average Composition.

The estimation of the average composition of igneous rocks as a whole is not such a simple matter as may be thought at first, because of several complicating factors which should be taken into account, and certain corrections in some of our data which should be made, to obtain fairly satisfactory results. We are not yet in a position to make precise allowance for these, into the discussion of which we cannot enter here, so that the results so far obtained can be regarded as but first approximations, only roughly correct but of some value.

The most recent and reliable calculations are three made successively by Prof. F. W. Clarke, of the United States Geological Survey, two of the igneous rocks of the British Isles by Prof. A. Harker, of Cambridge, and one made by myself some years ago.[2]

Clarke's latest estimate is based on more than a thousand analyses of igneous rocks of the United States made by the chemists of the Geological Survey, and mine is based on about 1,800 analyses of igneous rocks from all parts of the globe, and made by many analysts of various nationalities. These are shown respectively in columns I. and II. of Table I., only the more important constituents being considered, and the whole being reduced to 100 per cent. Harker's estimates are omitted, but in general they conform to those here presented.

It is evident that the two are very closely alike, the only noteworthy divergence being in the amount of silica. The

[2] F. W. Clarke, *Bulletin of the U. S. Geological Survey*, No. 148, p. 12 (1897); No. 168, p. 14 (1900); No. 228, p. 18 (1904); A. Harker, *Geological Magazine*, vol. xxxvi., p. 18 (1899); Tert. Ign. Rocks of Skye, p. 416 (1904); H. S. Washington, *Professional Paper* No. 14, *U. S. Geological Survey*, p. 108 (1903). I am at present engaged in new calculations of the average rock, based on more than 3,000 analyses, but this is not yet ready for publication.

TABLE I.—*Average Composition of Igneous Rocks.*

	I. United States, Clarke, 1904.	II. The World, Washington, 1903.	III. Complete Average, Clarke, 1904.		IV. Clarke, 1904.
SiO_2	60.48	57.78	59.87	O	47.09
Al_2O_3	15.17	15.67	15.02	Si	28.23
Fe_2O_3	2.61	3.31	2.58 +	Al	7.99
FeO	3.44	3.84	3.40 +	Fe	4.46
MgO	4.10	3.81	4.06	Ca	3.43
CaO	4.84	5.18	4.79	Na	2.53
Na_2O	3.43	3.88	3.39	Mg	2.46
K_2O	2.96	3.13	2.93	K	2.44
H_2O (110° +)	1.48	1.42	1.46	Ti	0.43
H_2O (110° —)	0.41	0.36	0.40	H	0.17
TiO_2	0.72	1.03	0.72	C	0.14 —
P_2O_5	0.26	0.37	0.26	P	0.11
MnO	0.10	0.22	0.10	S	0.11
CO_2	.....	.....	0.52 —	Ba	0.089
S	.....	.....	0.11	Mn	0.084
BaO	.....	.....	0.11	Cl	0.07 —
Cl	.....	.....	0.07	Sr	0.034
Cr_2O_3	.....	.....	0.05 —	Cr	0.034 —
SrO	.....	.....	0.04	Zr	0.026 —
ZrO_2	.....	.....	0.03 —	Ni	0.023 —
NiO	.....	.....	0.03 —	F	0.02 +
V_2O_3	.....	.....	0.03 —	V	0.02 —
F	.....	.....	0.02 +	Li	0.01
Li_2O	.....	.....	0.01		100.00
	100.00	100.00	100.00		

higher figure in I. may be ascribed in part, as pointed out by Clarke, to the inclusion of many separate silica-determinations, which were undertaken on rocks comparatively high in this constituent; in part to the inclusion in the estimate of some analyses of siliceous gneisses and schists, rocks which were not regarded in my estimate; and probably in part to the fact that in Clarke's estimate only rocks from the United States were considered. But notwithstanding the slight discrepancies and the uncertainty introduced by non-allowance for the disturbing factors mentioned above, the results of the two calculations are so concordant that we may feel a high degree of confidence in the belief that the data given here approximate closely to the average composition of the earth's accessible igneous rocks.

In column III. are given the results of a more complete estimate by Clarke, which includes the minor constituents frequently present in igneous rocks, but which are only deter-

mined in the more complete analyses, as those made by the chemists of the U. S. Geological Survey. It will be seen that the data are in agreement with the general composition of the most common rock-forming minerals, their constituents, silica, alumina, ferric and ferrous oxides, magnesia, lime, soda, potash, and water, making up 97.9 per cent.

The estimated amount of carbon dioxide is undoubtedly too high, and is due to the number of analyses of altered rocks which were included in the estimates.

We may examine the matter further and, resolving the oxides into their elementary components, ascertain the average amounts of the elements in the igneous crust of the earth. This problem has been studied by Clarke in the papers already cited, and by Vogt[3] in Norway. The results of Clarke's latest calculations are given in column IV. of Table I., the figures including those for the minor constituents of column III., just noticed. In an earlier computation Clarke introduced estimates of the elements which make up the air, the water of the oceans, and such non-igneous rocks as limestone and coal. But the introduction of these into the calculation does not materially alter the final results from those given here, in which they are omitted, since these bodies are of relatively very slight quantitative importance compared with the whole mass of known rocks, however large they may loom to our eyes. Ore-bodies also are quite negligible in this connection.

These data show that oxygen composes almost one-half, silicon more than one-quarter, and aluminum about one-twelfth of the earth's crust (the three together amounting to 83.3 per cent.), while iron, calcium, sodium, magnesium, potassium, and titanium follow after in the order named in decreasingly small amounts. Thus, only nine elements together constitute 99 per cent. of the igneous crust. This is certainly a very remarkable fact, and one doubtless of great significance for the proper understanding of the true constitution of our globe, could we but interpret it aright, as some day we may hope to do.

The relatively high position occupied by titanium, ninth on the list in the order of abundance, is also a striking feature, as this element is commonly supposed to be rare. The establishment

[3] J. H. L. Vogt, *Zeitschrift für praktische Geologie*, pp. 225–238, 314–325 (July and September, 1898).

of this fact is largely due to the accuracy and completeness of the rock-analyses made by the chemists of the U. S. Geological Survey, and its great abundance was unsuspected before they began their long series of excellent analyses, though its wide distribution had been noted.

It is also noteworthy that, with the exception of iron, aluminum, manganese, and nickel, none of the metals commonly used as such appear in the list, while others, which are of very limited practical application, are present. While nearly, if not quite, all of the elements are presumably present in igneous rocks, the average amounts of those not given in the list are so extremely small that they may be regarded as minor corrections to be applied in the future to certain of those here given, since nearly all of them would be precipitated and weighed in the course of analysis with some of those more abundant.

In the important paper cited above, Vogt has discussed the probable amounts of these missing elements, and a brief statement of those of his results which pertain to the more important metallurgical elements may be given. The estimates, it must be premised, are but approximations, and only indicate the magnitude of the several amounts as percentages of the earth's crust. But they serve to show the average extremely small quantities of many metals and other elements which are usually regarded as quite common or at least not very rare.

The percentage amounts of tin, zinc, and lead are expressed by a digit in the third or fourth decimal-place, that of copper in the fourth or fifth, that of silver in the sixth or seventh, that of gold in the seventh or eighth, the amount of platinum being about the same. Mercury is rather more abundant than silver, and arsenic, antimony, molybdenum, and tungsten less than copper and greater than silver; while bismuth, selenium, and tellurium are less abundant than silver but more so than gold.

III. Petrographic Provinces.

More than 30 years ago, Vogelsang[4] pointed out that the igneous rocks of certain districts—called by him *geognostische Bezirke*—showed certain textural or mineral characters in common, which served to distinguish them from the rocks of other

[4] H. Vogelsang, *Zeitschrift der deutschen geologischen Gesellschaft*, vol. xxiv., No. 3, p. 525 (May–July, 1872).

districts. The same idea was expressed later by Judd,[5] who introduced the term Petrographic Province, and was afterward elaborated by Iddings,[6] who likened the districts of similar rocks to families and referred to their relationships as "Consanguinity." Neither Judd nor Iddings seems to have been aware of Vogelsang's prior publication. The first two statements referred only to geological occurrence and to textural and mineralogical peculiarities; while Iddings, writing at a time when the chemistry of rocks had begun to assume its due prominence (largely owing to the earnest labors of the chemists of the U. S. Geological Survey), showed that the relationship is also indicated by the chemical composition of the various rocks, and is fundamentally dependent on this, and he consequently devotes much space to the chemical evidence of consanguinity.

The doctrine of consanguinity, as it may be termed, has now received general acceptance, and it is commonly recognized that the igneous areas of the earth's surface are divisible into districts the rocks of which show certain features in common which serve to distinguish them from those of other districts. It is also assumed that the possession of these common features, especially those dependent on the chemical composition, indicates that the rocks of a given district have a common genetic origin, that is, are derived from a common parent body of magma. The processes by which this differentiation and derivation from a common magma are brought about are still obscure, and form the subject of much modern investigation and discussion, into which we cannot enter here. Such areas of related rocks are usually called petrographic provinces, though the term comagmatic region, which indicates more clearly their derivation from a common magma, has recently been introduced.[7]

Though petrographic provinces represent one of the prominent phases of the distribution of the elements in the earth's crust, and though their existence is, in general, undeniable, yet their characters are so complex and made up of so many factors that the characterization in most cases cannot be made very

[5] J. W. Judd, *Quarterly Journal of the Geological Society*, vol. xlii., pt. 1, No. 165, p. 54 (Feb. 1, 1886).

[6] J. P. Iddings, *Bulletin of the Philosophical Society of Washington*, vol. xii., pp. 128–144 (1892).

[7] H. S. Washington, *Carnegie Publication* No. 57, p. 5 (1906).

precise or the limits be very sharply drawn. Their study is still almost in its infancy, and the accumulation of many more data, especially from the analytical side, is needed before definitive studies can be undertaken, and the characters of any petrographic province be precisely defined.

The geological evidence of consanguinity is at times clear. Thus, as Iddings[8] says:

"The constant recurrence of particular series of rocks, often with a certain order of eruption in different localities, and the frequent occurrence of such series at neighboring centers of volcanic activity, would be enough to justify the belief that there was a definite connection between the members of a group."

On the other hand, cases are known where such geological evidence is lacking or conflicting, or where the relations are so generally observed as to be meaningless in this connection.

Geologically speaking, a petrographic province may belong to any period of geologic time, or may conceivably extend over more than one period. The region may be small or large, covering hundreds or hundreds of thousands of square miles. It may be of any shape, forming an elongated band or zone, a highly irregular area, or one more or less equidimensional. It may consist of a single, large area of connected intrusive or effusive rocks, or of adjacent but isolated areas.

The chemical characters which, being common to the rocks of a province, indicate consanguinity are manifold. The rocks may be uniformly high in soda or in potash, or in potash and lime, low in magnesia and high in iron, generally deficient in silica, and so on. Throughout one province the soda may increase relatively to potash as silica decreases, while in another the reverse holds good. Or again, there may be some combination of such two kinds of characters, called respectively absolute and serial. The subject is complicated by the possibility of local differentiation, so that in a region of unquestionably related rocks, we may meet with some the character of which does not conform to the general law of the region, but the presence of which is to be explained by the extreme differentiation of some portion of the general magma.

Conforming to the chemical features, and in general largely dependent on them, are the common mineralogical characters.

[8] *Loc. cit.*, p. 43.

These are very important, not so much in themselves, as because they often enable one practically to determine the general relationship without the necessity of a long series of analyses. The mineralogical similarity may be evident in two ways: either by the general presence of certain minerals which are rare or are not usually found in certain kinds of rock, as the rare zirconium minerals, or nephelite, or leucite, the occurrence of hypersthene in the basalts and andesites, or of biotite in peridotites; or by certain peculiarities of color, form, or other characters shown among the more usual mineral groups, and which are dependent on the introduction of certain chemical constituents into the molecule, as bright green, pleochroic augites or blue hornblendes, due to their containing much soda, purple augites or red-brown hornblendes, due to titanium, and so on. Here again the possibilities of difference are numerous, but the mineralogical evidence of relationship is often so marked as to be unmistakable to the petrographer.

A good illustration of such petrographic provinces and of their distribution is furnished by the United States, which may be briefly described, though our knowledge is still incomplete. Stretching along and rather close to the Atlantic coast is a zone of small, isolated areas of igneous rocks which are chiefly characterized by a high content in soda, resulting in the common presence of nephelite-syenites and other rocks containing nephelite, peculiar hornblendes, and other minerals characteristic of such magmas. This zone includes areas in Quebec, New England, New Jersey, Arkansas, Texas, extends into eastern Mexico and the West Indies, and probably as far south as Brazil and Paraguay. Parallel with this, but usually farther inland, is a second zone of areas of rocks which are low in silica and the alkalies, but high in lime and iron. This starts in the great anorthosite area of eastern Canada and Labrador, appears in New England, the Adirondacks, Delaware, Maryland, and extends to Georgia and possibly farther south. The rocks of this region are typically gabbros, diabases, and pyroxenites, dunite and other peridotites, with some granites high in lime, and are often accompanied by very basic ores and other products of differentiation which are very rich in iron and titanium. Farther inland and west of the Appalachian range is another belt, less well defined but apparently in

general parallel to the others, of widely isolated small occurrences of peculiar peridotites and other rocks, low in silica and very high in magnesia and iron, with little lime or soda but much potash. This last feature gives rise to the common presence of peculiar micas, which distinguish these peridotites mineralogically from those of the preceding region. These areas occur in Quebec, New York, Pennsylvania, Kentucky, Arkansas, and probably still farther south.

Passing over the broad central part of the continent, where igneous rocks are very sparingly present, we find a province east of the Rocky mountains which is characterized by high alkalies, especially potash, so that the usually rare mineral leucite is here quite common. This region is best known in Montana, Wyoming, and Colorado, and may possibly extend into western Texas. In the region of the Rocky mountains and the cordilleras generally the occurrences of igneous rocks are so numerous and the relations so complex that it is somewhat difficult to unravel the various petrographic provinces. As a whole, however, the igneous rocks of this part of the continent seem to belong to one very extensive province, which is continued into Alaska on the north and along the Andes to the south. In general chemical character the rocks show rather low alkalies, with more soda than potash, rather high lime, and but moderate amounts of iron and magnesia, leading to the abundance of such ordinary rocks as feldspar-basalts, andesites, dacites, and some rhyolites. There is some evidence that the province as a whole may be divisible into several subordinate districts, but it is noteworthy that rocks so high in soda or potash as to contain nephelite or leucite are practically unknown west of the Rocky mountains. There are also indications of what may be a distinct region along the coast ranges which is characterized by high soda and generally high silica, but this demands further investigation.

A number of petrographic provinces outside of the United States may also be briefly indicated. One of the best known is that of southern Norway, which is prominent through the classic researches of Brögger, the rocks of which are characterized by high alkalies, especially soda, and the presence of many minerals elsewhere rare. This is possibly connected with the region of the Kola peninsula in northern Finland. The

British Islands, with the Faeroes, Iceland, and probably Spitzbergen, form another well-defined province, the rocks of which resemble on the whole those of our Rocky Mountain region, though they differ in some respects. Leaving aside Germany, Austria, and France, each of which contains several petrographic provinces, the relations of which appear to be somewhat complex, in the basin of the Mediterranean we find at least three well-defined and quite distinct provinces. In the eastern part, including the Grecian archipelago and parts of Asia Minor, the rocks are rather siliceous, with fairly high lime and rather low alkalies, soda dominating potash, so that dacites, andesites, and feldspar basalts are prominent. Hypersthene is here rather common. The Italian peninsula shows a second, very well-defined province, which embraces seven distinct volcanic centers along the west coast. The rocks of this are remarkable for their high content in potash, which at times reaches extraordinary figures, and leads to the abundant presence here of the mineral leucite, which is elsewhere decidedly rare. Lime is also rather high, while soda, iron, and magnesia are low. The other provinces of continental Italy have not been thoroughly studied and are less well known. In the western basin of the Mediterranean, including localities in Spain, Sardinia, some islands south of Sicily, and probably southern France, there appears to be a third province, which differs from the others in that soda is much higher and the more basic rocks (basalts) contain very large amounts of titanium, and in other ways. This last may be connected with a province in eastern Africa, running down the Great Rift valley and including parts of Madagascar, in which rocks rich in soda are very common. A somewhat similar province appears to exist in New South Wales and Queensland in Australia.

The descriptions just given, which are but the barest sketches of only some of those which are known, and with no references to authorities, will serve to give an idea of some of the leading distinguishing features of petrographic provinces, and how multifariously they are scattered over the earth's surface. Their existence and distribution indicate clearly that the underlying magma basins, or the sources from which the igneous rocks are immediately derived, are not everywhere uniform and alike, but that there exists a certain heterogeneity in the

non-sedimentary parts of the earth's crust. It should, however, be noted that two provinces, though widely separated, may be essentially alike in all their features, as is the case with that of the eastern Mediterranean and that which extends from the Andes to Alaska.

IV. The Correlation of the Elements.

The existence of petrographic provinces is a broad phase of the distribution of the elements among igneous rocks, the distribution being essentially a spacial one and the evidence resting almost entirely on the relative proportions of the most abundant elements. But apart from this spacial distribution there is evident, also, a correlated variation among the elements; that is, a tendency for certain ones to increase or decrease, to be relatively abundant or not, according to the presence or absence of others. The causes of this behavior are obscure and apparently complex. In part they may be probably referred to certain fundamental relations among the elements, as shown by the periodic classification and chemical affinity; in part to the effect of certain physico-chemical laws leading to the mutual segregation of elements affected similarly; and possibly in part to the degradation of some of the elements, as indicated by recent experiments of Ramsay. But any discussion of the causes is outside the province of this paper, in which we can only deal briefly with some of the facts of distribution.

The study of these mutual relations among the elements in igneous rocks is of recent date, and has been made possible, especially so far as the rarer elements go, only by the completeness and accuracy of modern chemical analyses of rocks. Such analyses supplement the evidence afforded by study of minerals, mineralogical associations, and ore-deposits, and, dealing as they do with what must be regarded as the ultimate source of the ores, are of the highest significance and importance. In the following brief discussion, therefore, stress will be laid on the evidence afforded by rock-analyses, with some reference to chemical mineralogy, while ore-deposits, as being more technical and better known to the mining engineer, will be alluded to only occasionally.

Considering first only the most abundant elements, a study

of the igneous rocks in general shows that silica, alumina, soda, and potash tend to increase or decrease together, though not always at the same rate; while, on the other hand, the iron oxides, magnesia, and lime tend to vary together and in general inversely as the preceding constituents. The more siliceous rocks almost invariably show relatively high alumina and alkalies and low iron oxides, magnesia, and lime, leading to the common presence in abundance of the alkali feldspars and the comparative paucity in calcic feldspars and the ferromagnesian minerals, which tend to increase rapidly with diminution in the silica-content. Highly siliceous rocks which contain more iron, magnesia, and lime than alumina, soda, and potash are of very exceptional occurrence. The rule mentioned above is so generally true that it may be regarded as the normal one for igneous rocks in general, and is commonly accepted as such in petrology.

At the same time, there is considerable evidence that certain subsidiary relations obtain among the constituents other than silica, which, while by no means universal, are at times very pronounced, and occasionally seem to supersede the more general law. Thus, soda not uncommonly tends to vary with the iron oxides, while potash shows similar relations to magnesia, resulting in the presence of potassium minerals in highly magnesian rocks and the abundance of sodium minerals in those high in iron. Again, while neither iron nor magnesia shows any marked affinity towards or tendency to vary with alumina, this constituent and lime are occasionally found to occur together in great abundance and to the general exclusion of the others. These relations are also evident in certain facts of chemical mineralogy, as the usual predominance of magnesium over iron in the potassic biotites and phlogopites, the abundance of soda and absence of potash among the highly ferriferous augites and hornblendes, and the numerous silico-aluminates containing much calcium, while those with iron or magnesium are comparatively very rare.

But this tendency to selective and correlated variation among the chemical constituents of igneous rocks is not confined to those which are present in greatest amount. It is equally well, and indeed in some respects more strikingly, shown among the rarer elements, both as compared with those which are most

abundant and with each other. Furthermore, the distribution of some of these rare elements would seem to have important bearings on some of the problems of economic geology and the distribution of ore-deposits.

The general facts of this distribution and variation of the rare elements have been summarized in several recent publications,[9] but many of the details are still un-coördinated and widely scattered through the vast mass of petrographic literature, and there are certain aspects and recent developments which are either neglected, or only briefly alluded to, in the publications cited.

It is now commonly understood that certain elements are prone to occur most often and in largest amounts in rocks which are high in silica, the so-called "acid" rocks; while others are met with similarly in those low in silica—the "basic" rocks. This is essentially the only set of relations recognized by Vogt, while De Launay in addition to these two groups proposes two others, the "mineralizing agents" and the vein-metals.

Evidence, however, is accumulating that the relations of the rare elements to the igneous rocks cannot be expressed so simply as is done by Vogt and De Launay. Their relative abundance is not dependent on the silica alone, and hence referable only to the "acidity" or "basicity" of the rock. The relations are more complex, and dependent, not so much on the amount of silica, as on the relative amounts of other constituents, notably soda, potash, iron, magnesia, or lime. They conform, on the whole, to the general relations of the most abundant constituents, some of the rarer elements being characteristically at home in the rock groups which show high alumina and alkalies, and which include those high in silica; while others again are most abundant in the rocks high in iron, magnesia, or lime, and which consequently most often show low silica percentages. Further than this, on the one hand, certain rare elements are not equally at home in the alkalic

[9] J. H. L. Vogt, *Zeitschrift für praktische Geologie*, p. 326 (September, 1898); J. F. Kemp, *Ore Deposits*, 3d edition, pp. 34–37 (1900); H. S. Washington, *Manual of the Chemical Analysis of Rocks*, p. 14 (1904); L. De Launay, *La Science Géologique*; p. 637 (1905); W. F. Hillebrand, *Bulletin No.* 305, *U. S. Geological Survey*, p. 21 (1907).

rocks in general, but are most abundant either in those high in soda or in those high in potash. On the other hand, some of the elements segregated in the basic rocks seem to be most at home in those which are highly calcic, others in those which are high in iron or in magnesia, though here the evidence is not so clear and the distinctions apparently not so well marked as in the preceding case.

We may consider first those minor constituents of rocks which are determined in the most modern and complete analyses, and next those which exist in rocks in such small amount as almost to defy determination by the usual analytical methods, but whose presence is made known either mineralogically or by their segregation in ore-deposits. The second group includes almost all of the commercially important metals (except iron and manganese), while the former includes many elements which are assuming an increased practical importance as their economic possibilities and uses become better known. In a general way the elements will be taken up in the order of their positions in the periodic classification. No references will be given, as an attempt to render them complete would unduly lengthen the paper. This course seems the more advisable, in spite of the apparent injustice to those whose invaluable work and contributions must thus be ignored, since the present paper may be considered as merely a preliminary one to a more exhaustive and monographic treatment which it is hoped to publish later.

Lithium is very widely distributed among igneous rocks, but always in very small amounts. While it frequently is to be detected by the spectroscope, it seldom occurs in weighable quantities, and the difficulty of its exact separation from the other alkali metals and its comparative unimportance cause it to be but seldom estimated quantitatively. Although such minute traces are present in both acid and basic rocks, yet it is undoubtedly more closely connected with those which are highly siliceous and alkalic. The minerals in which it forms an essential component, as spodumene, lepidolite, amblygonite, and some tourmalines, are most often met with in granites and pegmatites derived from granitic magmas. Unfortunately, the granites and pegmatites which carry lithium minerals most prominently do not appear to have been analyzed,

but there is reason for the belief that they are sodic rather than potassic in general character. The very common association of lithium with soda rather than with potash in many minerals also points to the same conclusion.

Beryllium is much like lithium in its associations, beryl and other rarer minerals which contain it occurring for the most part in granites or pegmatites. Few analyses exist of such beryl-bearing rocks, and beryllia has seldom been estimated separately from alumina in rock-analysis, but such data as are available and the common mineralogical association of beryllium and sodium point to the conclusion that the element is most at home in sodic magmas.

Attention may be called to the fact that beryl, in spite of its common occurrence, is not given in the list of descriptions of the rock-forming minerals, such as those in the standard works of Zirkel, Rosenbusch, and Iddings, though Lévy and Lacroix briefly described it in their work[10] and it is placed on their large colored table of bi-refringences. In its optical properties it closely resembles nephelite and apatite, and being hexagonal in crystallization as well, might readily be mistaken for one of these minerals. I have noted the fact that analyses of nephelite-syenites and other highly sodic rocks frequently show a decided excess of alumina which cannot be explained by the apparent mineralogical composition of the rock, and the suggestion is made that this is possibly due to the presence of beryl, the beryllia of which would appear as alumina in the course of analysis, unless special means were taken to separate the two. On the other hand, the excess of alumina may be real and due to the composition of the magma.

Strontium has been shown by the analyses of the U. S. Geological Survey to be widely distributed in the rocks of this country. I have found it almost invariably when looked for in many European rocks, and it is almost constantly present in those of Australia. But it seldom occurs in more than traces, and the evidence in regard to its distribution is as yet inconclusive, in spite of the many modern analyses in which it is now determined, chiefly because of the small amounts usually met with. It would appear to be most abundant in rocks somewhat high in lime and with moderate to rather low silica,

[10] *Les Minéraux des Roches* (Paris, 1888).

though it is worthy of note that the highest figures recorded for it are in some rocks of Wyoming which are low in lime and extraordinarily high in potassium and barium. Being but rarely a constituent of silicate minerals, decisive evidence from this side is wanting, though it occurs with lime in some heulandite and brewsterite.

Barium is another element which the analyses of the Washington chemists showed to be widely distributed, and almost invariably in decidedly greater amounts than strontium. It is now often determined by analyses of superior quality, and in a recent study, embracing the rocks of Italy, the United States, and New South Wales, I have shown that it is specially prone to occur in potassic rocks, sometimes when the potash is accompanied by considerable lime, but that it is rarely met with in notable amount in decidedly sodic or calcic rocks. Neither the amount of silica nor the relative proportions of iron and magnesia, appears to be a determining factor of much importance. This association of barium and potassium in igneous rocks is in harmony with the mineralogical evidence. Barium is a frequent minor constituent in potassium minerals, as orthoclase, muscovite, and biotite, while potassium accompanies barium in hyalophane and harmotome. On the other hand, barium is not reported in analyses of sodium minerals, but occurs in small amounts in the calcium zeolites, brewsterite and phillipsite. Barium also seems to tend to associate with manganese, as shown by its common occurrence in psilomelane, and the occurrence of minerals of the two metals in certain mines.

Boron is seldom or never mentioned in rock-analyses, chiefly because of the complexity and difficulty of its exact determination, especially in very small amounts. But it is not infreqently present in weighable amounts in granites and pegmatites, chiefly as a constituent of tourmaline. The few analytical data that we have of such tourmaline-bearing rocks are not decisive, but boron does not appear to have very decided preferences for either soda or potash. Its associations in minerals are likewise not strongly marked, but among the silicates calcium is the basic element which most frequently accompanies it, and soda is more commonly met with in boron-bearing minerals than is potash. Boron is commonly regarded as one of the pneumatolytic elements.

Cerium, *yttrium*, and the other metals of the so-called "rare earths," as well as *thorium* and *uranium*, are only rarely determined in rock-analysis. Minerals containing them are commonly associated with acid pegmatites, which, judging from occurrences in Norway, Greenland, and elsewhere, are most apt to be sodic, though the few determinations available of the rare earths are in highly potassic igneous rocks.

Titanium is, as we have seen, far from being the rare element which it was formerly considered, and it is probably never wholly absent from any rock. It is distinctly much more abundant in basic than in acid rocks, and its affinities in the magma seem to be decidedly rather with iron than with magnesium, and still less with lime. While it is not commonly associated with alkalic rocks, yet when these are low in silica it shows a tendency to be present in considerable amount when the rock is sodic, as indicated by recent rock-analyses; and this tendency to association of titanium with sodium appears mineralogically, as in the soda-amphiboles, some of which are highly titaniferous, and in certain rare minerals, as astrophyllite and rosenbuschite. Highly potassic and highly calcic rocks seldom show large amounts of titanium, though most of the mineral titanates contain calcium as the base.

Zirconium, so closely allied to titanium chemically, also shows certain analogies in its magmatic relations. While unlike titanium in being rare in the basic rocks, those high in iron, magnesia, and lime, and referred by Vogt to the acid rocks, presumably because of the common occurrence of zircons in granites, it is now commonly recognized by petrographers that zirconium is by far most abundant in the rocks which are high in soda. Indeed, zirconium may be considered to be a characteristic minor chemical constituent of the sodic rocks, whether the silica be so high that quartz is present, or whether it be so low that nephelite is abundant, as in the nephelite-syenites and phonolites. Practically all modern, complete analyses bear out this view, which is confirmed by the common association of sodium and zirconium mineralogically, as in eudialyte, catapleiite, wöhlerite, and the zirconium pyroxenes.

Phosphorus, as a constituent of apatite, is universally diffused in small amounts through igneous rocks, and is most abundant in the basic ones, though its relations to the constituents

other than silica are not clear. Study of large collections of analyses indicates that it is usually, but not always, associated with high lime, rather than with high iron or magnesia, and in some distinctly sodic provinces the more basic rocks with high soda and abundant nephelite show high figures for phosphorus pentoxide. Some of the phosphates are met with in granitic and syenitic pegmatites.

Vanadium has been shown by the researches of Hillebrand to be quite widely distributed, but always in very small amount and almost wholly confined to the basic rocks. As it exists as the sesquioxide, V_2O_3, replacing alumina and ferric oxide in ferro-magnesian minerals, it is especially abundant in rocks composed largely of pyroxene, hornblende, or biotite, while it is present only in traces or not at all in rocks very rich in olivine, where the iron is present mostly as ferrous oxide, as in the peridotites. It is associated with iron rather than with magnesium, and occurs in most abundance in some iron-ores of magmatic origin, but no definite relations to the alkalies can be made out. Its common occurrence in ashes of coals and its abundance in certain carbonaceous deposits recently described are noteworthy, though outside the present discussion.

Sulphur is, by far, more abundant in the basic than in the siliceous rocks. It may exist, in the oxidized condition, in the minerals hauynite and noselite, in which case the rocks containing these minerals are almost invariably distinctly sodic; or it may form sulphides, as pyrite, pyrrhotite, and chalcopyrite, these being most common in rocks rather high in iron, magnesia, and lime.

Chromium, like vanadium, is a constituent of the basic rocks, but, unlike this, is most abundant when magnesia and not iron is high, and when olivine, rather than pyroxene or hornblende, is abundant, in spite of the fact that it occurs as the sesquioxide, Cr_2O_3. Presumably this is because, instead of replacing alumina and ferric oxide in the ferromagnesian minerals, it is most commonly met with in the minerals chromite and picotite. It is reported to reach very high figures again in certain effusive rocks which are so high in lime and low in silica that the rare mineral melilite is present.

Molybdenum is seldom looked for in rock-analysis, and our knowledge of its magmatic relations is based almost wholly on

an investigation of Hillebrand. He found that it is much less common and is present in smaller quantity than vanadium, and that, unlike the latter, it is present only in the more siliceous rocks, though in quantities too small to permit of further discrimination. As molybdenite it occurs most often in quartzose rocks.

Fluorine is almost universally present in very small amount as a constituent of most apatites, and is usually regarded as a "mineralizing agent," and, as such, is frequently present in pneumatolytic minerals. As stated by Vogt, it seems to be more common in the acid rocks, but there seems to be a marked tendency on its part to favor especially rocks which are high in soda. This is seen in the fact that fluorite is frequently present as an original constituent of such highly sodic rocks as nephelite-syenite, phonolite, and tinguaite; the association of fluorine and sodium in certain rare minerals, as leucophanite, meliphanite, johnstrupite, rinkite, etc., which are almost always found in sodic rocks; and by the recent discovery by Lacroix of sodium fluoride in nephelite-syenites of West Africa.

Chlorine resembles fluorine in being a pneumatolytic constituent, and is present in igneous rocks chiefly in the minerals sodalite and noselite, which are almost wholly confined to sodic rocks and especially those which are low in silica, in this resembling the occurrence of SO_3.

Manganese, though as widely distributed as titanium and phosphorus, is usually present in such small amounts as not to allow a clear judgment of its magmatic relations, especially as the high figures often reported for it are apt to be due to analytical error. As a general rule, its amount is greater in the basic rocks, and certain considerations indicate a preference for rocks high in iron rather than in magnesia or lime, but the variations are not very significant. We have already noted above the tendency to association of barium and manganese.

Nickel is pre-eminently at home in the basic rocks, especially in the peridotites and serpentines, where it replaces iron in the olivine, while it likewise occurs in small amounts in hornblende, biotite, and in pyrite and pyrrhotite. Certain rather high figures reported for it may be ascribed to analytical confusion with platinum derived from the utensils employed; but

researches now in progress indicate that it may be present in considerable amount (up to about 0.20 per cent.), not only in the "basic" but in the more siliceous rocks of certain localities where its presence has not hitherto been suspected. It is reasonable to suppose that it is most apt to be present in rocks which are relatively high in iron rather than in magnesia and lime, and the results of the investigation just mentioned are in harmony with this supposition.

Cobalt almost always accompanies nickel in igneous rocks, but always in extremely small and scarcely weighable amounts.

The elements belonging to the next group to be discussed, those which are scarcely detectable in igneous rocks by the usual analytical methods on account of the excessively minute amounts usually present, need not detain us long, even though they are commercially among the most important. Since the analytical data are either very scanty, untrustworthy, or wanting altogether, and their presence is revealed to us mostly through secondary processes of concentration in veins, placers, and other ore-deposits, we are not yet in a position to generalize with confidence as to the magmatic relations of most of them.

Furthermore, having but slight affinity for silica, and thus (with few exceptions) seldom forming silicates or entering as minor constituents into the silicate minerals of other elements, we are deprived to a very great extent of this kind of evidence.

Copper is not infrequently reported in analyses of igneous rocks, but, as pointed out by Hillebrand, its apparent presence may often be attributed to contamination during the course of analysis, or, as may be suggested here, to confusion with platinum likewise due to contamination, as was suggested in the case of nickel. But notwithstanding these sources of error copper seems to be widely distributed among igneous rocks, though in very small amounts. There seems to be little doubt that it is most abundant in the more basic rocks, especially those which carry pyroxene and hornblende rather than olivine, but no evidence seems to exist as to its relations to the chemical constituents other than silica.

Silver and *gold* have both been detected analytically in igneous rocks, while metallic gold has also been observed as an apparently primary ingredient of some rhyolites and granites. Both of these metals are "cosmopolitan in their relations," as

Kemp puts it, and they are known to occur in such highly siliceous rocks as granite, rhyolite, and quartz-porphyry, and, on the other hand, in diabase and gabbro. There is, however, good reason for the belief that gold, and probably silver as well, are most apt to occur in rocks high in silica, but their relations to the other elements are still quite unknown.

Zinc and *cadmium* (which latter is found only in connection with the former) are also very uncertain. There is, however, some reason for thinking that zinc is more apt to be present in acid rocks, as granites, this opinion being based on a few analytical data and the facts of some of its occurrences. The common occurrence of zinc in limestones, due presumably, at least in part, to the precipitating effect of the sedimentary rock, has no apparent bearing on its relations to igneous magmas.

Mercury is considered by G. F. Becker to be associated with granites, but his evidence is not very convincing. Its usual occurrence in sedimentary rocks tends to obscure its true relations, and, to the best of my knowledge, it has never been looked for or reported in an analysis of an igneous rock.

Tin, as the oxide, cassiterite, almost invariably occurs as the result of pneumatolytic processes in pegmatites, granites, and other rocks high in silica, and the mineral has been found in some rhyolites. Judging from the common association of cassiterite with lithium and beryllium minerals, and the presence of small amounts of tin in certain feldspars, micas, zircons, and in the rare mineral stokesite, it is probable that tin is associated rather with distinctly sodic than with potassic or calcic magmas, but much more chemical study of the rocks in which it occurs is needed to elucidate its relations.

Lead can often be found in rocks by using large amounts of material, and is occasionally reported, as in the analyses of rocks from British Guiana by J. B. Harrison and in those of rocks from New South Wales. No generalization in regard to it is possible as yet, but I am inclined to think that, like zinc, it favors the acid rather than the basic rocks, though it has been observed in both. The remarks in regard to the occurrence of zinc in limestones apply as well to lead.

Platinum and the other metals of this group are, as is well known, most commonly found in connection with peridotites, rocks low in silica and high in magnesia, though it has been

observed by Kemp in gabbros, which were presumably connected genetically with peridotitic rocks. Recent developments point to a somewhat wider distribution than was formerly thought to be the case, and indicate that platinum not infrequently is associated with copper-ores.

The true relations of such elements as *arsenic*, *antimony*, *bismuth*, *selenium* and *tellurium* to igneous magmas are quite unknown. It is possible that arsenic and selenium are most at home in the basic rocks, while antimony, bismuth, and tellurium are more apt to occur in siliceous ones.

We may summarize the observations recorded above as follows: Of the rarer elements whose distribution is better known, lithium, beryllium, cerium and yttrium, zirconium, uranium, thorium, sulphur (as trioxide), fluorine, chlorine, and possibly tin occur most abundantly in sodic magmas; barium in potassic magmas; titanium, vanadium, manganese, nickel, and cobalt in iron-rich magmas; chromium and platinum in magnesian magmas; and phosphorus (?) and chromium (?) in calcic magmas.

Of the other elements it can only be said that boron and molybdenum are certainly, and zinc, cadmium, lead, antimony, bismuth, and tellurium are possibly, connected with magmas high in silica; sulphur and copper almost certainly, and arsenic and selenium possibly, with those low in silica; while the relations of gold, silver, and mercury are very uncertain, but they are probably most at home in acid rocks.

This statement, it will be seen, differs from that of Vogt, in that, in the best-established cases, silica plays a less determinative rôle than some of the other major constituents. At the same time, the influence of the general law of the association of the most abundant oxides comes into play, and in a general way the potassic and sodic magmas are most apt to be highly siliceous (though the facts of distribution are shown in them even when silica is low); while those which are high in iron, magnesia, and lime are most apt to be low in silica.

Possibly the most striking feature of the distribution as thus shown is the great number of elements which are prone to occur in highly sodic magmas. As is well known, such magmas are those which show most tendency to differentiation and the formation of a great variety of rocks, many of them character-

ized by the presence of rare or otherwise unusual and interesting minerals, and there may probably be some connection between the two features of these magmas.

It will be noted that some of these elements, as fluorine, chlorine, sulphur (as trioxide), and boron, are among those to which is usually attributed the rôle of so-called "mineralizing agents," they being supposed to be present as dissolved vapors in the magma and to exert a marked effect on the crystallization of the mass, the formation of pegmatites, and so forth. It may be argued that such mineralizing and pneumatolytic elements are universally, and presumably quite uniformly, distributed among the rock-magmas, and that their presence in the highly siliceous and sodic rocks is due to the greater viscosity of these when molten, which would hinder the escape of gaseous constituents, while the basic magmas are more fluid at low temperatures and would hence allow such gases to escape before or during consolidation. On the other hand, it may be urged that the undeniable distribution, among magmas of distinctly different general chemical characters, of elements to which no such mineralizing or pneumatolytic rôle can be reasonably assigned, as barium, beryllium, zirconium, titanium, manganese, nickel, chromium, and platinum, would lead to the inference that the apparent distribution of the gaseous and "mineralizing" elements in igneous rocks is real and not dependent on physical causes. The subject is highly complex, and our knowledge of the fundamental facts and of the physicochemical laws involved is as yet inadequate for solution of the problem, further discussion of which would be outside the scope of this paper.

V. Practical Considerations.

When the facts of the relations of the occurrence of the rarer elements to the chemical characters of igneous magmas are considered, it is evident that their distribution over the earth's surface must be largely determined by that of the petrographic provinces. In other words, in any given petrographic province those rarer elements and minerals containing them would be most apt to occur abundantly which show a correlative tendency to association with the characteristic major constituents of the province. Thus, zirconium-bearing minerals and those of the "rare earths" should be most abundant in

provinces whose magmas are highly sodic and where such rocks as nephelite-syenite and phonolite are common; while chromium, nickel, and platinum would not be expected in these, but would rather be likely to occur in provinces where such rocks as gabbros and peridotites are the prevailing ones.

This idea has been recognized by Spurr[11] in his proposed term of "metallographic provinces," which is based largely on ore-associations, and which he applies more especially to those metals of most economic importance, such as gold, silver, copper, lead, and zinc. The probable very close connection between "petrographic" and "metallographic" provinces is pointed out by him, but the two classes seem to be regarded by him as distinct, at least to a certain extent.

When we deal with such complex bodies as veins and other ore-deposits, the matter is complicated by such factors as geological structure, the existence of faults, the occurrence of the igneous rock as plutonic masses, dikes or effusive flows, climatic conditions,[12] and other disturbing features. These may tend either to favor or to retard the processes of concentration which result in economically-exploitable metalliferous deposits. But these, though undoubtedly of the highest commercial importance, are subsidiary to the more fundamental facts of the distribution of the elements in igneous magmas, and it seems reasonable to suppose that a study of these latter features should be susceptible of results of great practical importance.

It seems that, at present, the knowledge gained by exact chemical analysis that the granites of a certain region contain minute traces of gold or of copper, would be of little use in guiding one in the search for the location of a gold- or a copper-mine. The prospector must always remain a valuable, indeed an invaluable, member of the mining fraternity. We cannot enter here into the vast and vexed subject of the genesis of ore-deposits, but if it were known by future researches that, for instance, gold or copper is normally associated with magmas of a certain general chemical character, a knowledge of this might conceivably be of material assistance in a search; not so much by

[11] *Trans.*, xxxiii., 336 (1903); *Professional Paper* No. 42, *U. S. Geological Survey*, p. 276 (1905); and No. 55, p. 128 (1906).

[12] H. V. Winchell, *Popular Science Monthly*, vol. lxxii., No. 6, pp. 534 to 542 (June, 1908).

indicating the exact position of a favorable location, but, in a more general way, by leading the prospector to confine his attention to a given region of favorable igneous rocks, and to disregard one whose rocks, on theoretical grounds, would probably result in loss of time and effort. Such a knowledge could be gained, not only by the complex and laborious methods of accurate and minutely complete chemical analysis, but more readily, at least in many conceivable cases, by simple petrographical examination and field-study of the most abundant and characteristic rock-minerals.

These considerations, it is true, are scarcely applicable as yet to search for such metals as gold, silver, or copper, concerning the magmatic relations of which our knowledge is of the vaguest description. But, in view of what has been ascertained by petrographical and chemical means of the distribution of other elements, it is not unreasonable to think that we shall eventually obtain well-founded and definite knowledge concerning the distribution of these also. Indeed, the opinion may be expressed that future petrographers will wonder at the fact that, for instance, the presence of such deep-seated and extensive deposits of copper as those at Butte, Mont., and in Shasta county, Cal., was so long unsuspected, and that their discovery came as a surprise.

At the present time a knowledge of the distribution of the elements is practically applicable, not so much to the metals of greatest human utility, as to certain elements the economic possibilities of which are only recently beginning to be appreciated as their chemical and physical properties and the application of these to commercial and economic purposes are becoming better known. Some illustrations may be permitted of the practical application of the facts set forth in the preceding pages.

If, for instance, one were in a new country or were engaged in a search for minerals containing such elements as zirconium, uranium, the rare earths, or beryllium, one would welcome a district of highly sodic igneous rocks, where albitic granites, nephelite-syenites, and phonolites were abundant; in this the chances of success would be most favorable. If the rocks were prevailingly gabbros, diabases, or feldspar-basalts one would reasonably assume that such minerals could not be expected to

occur, at least in such amount as to repay exploitation, and they would be neglected, or prospected for platinum or chromium, let us say. Similarly, if the platinum metals were found in the sands of a river the water-shed of which covered areas of gabbros, granites, and limestones, one would naturally turn to and explore the first in an attempt to trace the grains to their source, and would, with good reason, leave the others alone.

Instances of this kind could be multiplied, and, indeed, some present applications of the general principles are now practiced not infrequently, but without any suspicion of the true principles underlying them or realization of their more general applicability. Thus, in certain districts the occurrence of topaz or spodumene may be recognized as generally indicating the possible or probable presence of cassiterite, without appreciation of the more general and fundamental fact that the conjunction of tin, fluorine, and lithium is due to the distinctly sodic character of the igneous rocks.

With increase in our knowledge of the origin of ore-deposits, and a general agreement as to their ultimate source in igneous rocks (whatever be the divergence of views as to the processes of concentration), the probability of the future importance of such observations as have been outlined above, from a practical as well as from a theoretical standpoint, is fairly evident. We cannot as yet predict the probable presence of gold, silver, or copper in economic quantities from the petrographical and chemical study of the country-rock; but the time may come (and our increasing knowledge of igneous rocks justifies us in a certain degree of confidence that it will come) when such seemingly erudite and impractical studies will be able to guide us in certain regions, as to either the probable absence or presence of ore-bodies of such metals.

The problem is admittedly very complex, and is one which has not yet been studied enough to do much more than enable us to make a few broad guesses at the truth. But we are beginning to discern some glimmer of light, and the fact that we cannot make out clearly our guiding stars, veiled as they are by the mists of imperfect knowledge, should not cause us to disdain such help as glimpses of them may now afford, or underrate their possible importance when the mists shall have been dispelled.

The Air-Furnace Process of Preparing White Cast-Iron for the Malleablizing Process.

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(Chattanooga Meeting, October, 1908.)

THIS paper gives the composition of the iron and slag at different stages in the "air-furnace" process of preparing "hard metal," or white cast-iron, for conversion into malleable cast-iron by annealing. The details of the process are given in an appendix. One of us had been trying for years to work some chemist of some of our malleable-iron plants up to a degree of enthusiasm which would lead him to make such an investigation. It would have had the advantage of the greater familiarity with the conditions which one habitually on the spot must have. Failing in this, he made use of an opportunity which presented itself of taking samples here represented, with such fairness as the conditions permitted, holding firmly that a rough approximation to the truth is better than the complete lack of data, or that a bird in the hand is worth two in the bush. We hope that this attempt may stimulate others effectively to duplicate this work, and thus correct or confirm it.

Our results are shown in Table I. and graphically in Figs. 1 and 2.

Discussion.—At the time when the first two samples were taken, the metal and slag were still too sluggish to permit fair sampling, and not until sample 6 were the conditions really favorable, because at the time of the fourth sample the metal was not fully melted, and even at the time of sample 5 it was still sluggish. Indeed, the slag was probably very heterogeneous throughout, because it took up silica and alumina rapidly from the bottom, and because it was so siliceous as to be sluggish at this relatively low temperature. This heterogeneousness is exaggerated in the two samples of slag 8, apparently by the great viscosity due to its extraordinary acidity. Note that the manganous oxide of one sample is about two and

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TABLE I.—*Removal of the Foreign Elements in the "Air-Furnace."*

Sample No.	Time After Charging, 7.30 a.m.	Composition of Metal.					Composition of Slag.									Remarks.
		C.	Si.	Mn.	P.	S.	SiO_2.	Fe_2O_3.	FeO.	Al_2O_3.	CaO.	MgO.	MnO.	P_2O_5.	SO_3.	
	Hr. Min	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	
1	2–48	3.10	0.89	0.227	0.136	0.029	45.59	3.31	40.82	4.74	1.99	trace.	3.40	0.09	trace.	Charge not fully melted at throat; metal and slag sluggish.
2	3–03	3.12	0.89	0.227	0.137	0.029	61.54	3.27	25.50	5.18	0.75	trace.	3.69	0.11	trace.	
3	3–28	3.21	0.89	0.240	0.138	0.029	49.24	6.28	32.23	7.20	0.84	trace.	4.27	0.12	trace.	Not yet all melted at throat; almost no boiling.
4	3–45	3.16	0.89	0.227	0.136	0.029	57.67	2.95	27.94	7.25	0.84	trace.	3.39	0.16	trace.	Nearly all melted.
5	4–1	3.17	0.89	0.227	0.139	0.032	49.24	5.50	33.65	6.26	0.91	trace.	4.59	0.14	trace.	Nearly all melted—a little mushy. Slag now skimmed.
6	4–19	3.08	0.89	0.227	0.137	0.033	.....	.....	.....	.....	.....	.....	.....	.....	.....	Pours thin.
7	4–30	3.00	0.89	0.227	0.140	0.033	.....	.....	.....	.....	.....	.....	.....	.....	.....	First of melter's sample.
8	4–48	3.03	0.89	0.214	0.140	0.033	83.60 80.80	.....	4.94 6.45	9.49 10.00	0.83 2.30	trace.	0.92 0.38	0.18 0.21	trace.	Last of melter's sample. New slag formed spontaneously.
9	5–1	2.78	0.87	0.199	0.141	0.034	.....	.....	.....	.....	.....	.....	.....	.....	.....	Begin pouring 5 minutes after this.
10	5–17	2.71	0.87	0.195	0.141	0.035	.....	.....	.....	.....	.....	.....	.....	.....	.....	Charge nearly finished, as indicated by much sparkling. 5 minutes later, all out.

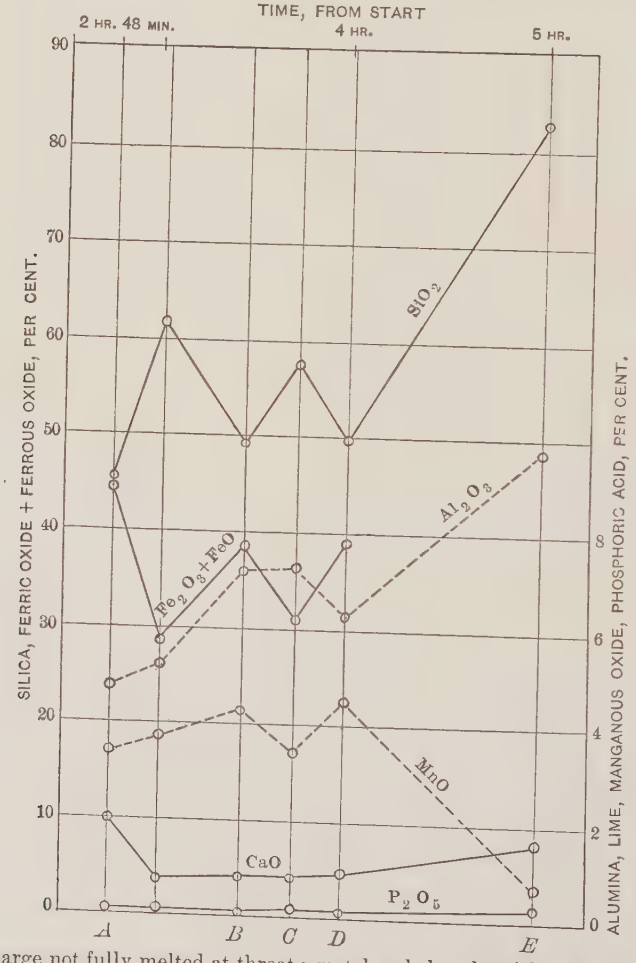
one-half times that of the other, and that the lime varies still more. Moreover, the great variations in the silica- and iron-oxide content of the other slags evidently represent rather the unevenness of the slag itself and the effect of additions of silica and alumina from the bottom, than the transfer of silicon from metal to slag and back. But the great change between slags 5 and 8 represents the fact that, immediately after taking sample 5 all the slag was skimmed off, so that slag 8 is a new one formed spontaneously, and evidently in chief part by the corrosion of the bottom.

Whatever allowance must be made for the heterogeneousness of the slag, the internal evidence goes to show that the sampling of the iron after the charge had been thoroughly melted was accurate. Note the slight but fairly regular increase of the phosphorus-content, representing the decrease of the weight of the charge by the slagging of iron; the still greater increase of the sulphur-content—greater when reckoned in percentage of the initial sulphur-content—representing this same slagging of iron, together with a slight absorption of sulphur from the furnace-gases; and the almost perfect uniformity of the silicon-content.

The increase of carbon between the samples 1 and 3 may be due to the progressive melting of the pig-iron. At the time when samples 1 and 2 were taken some iron remained unmelted. As this later melted it naturally changed the composition of the molten bath. Now the fact that this change was in the direction of increasing the carbon-content indicates that the loss of carbon in melting was greater in case of the iron which melted first than in that which melted later, perhaps because the former melted more slowly than the latter, as would naturally be the case. It is to be noted that, in the same way, sample 3 contains a little more manganese than samples 1 and 2.

The surprising thing in these results is the constancy of the silicon-content from sample 1 to sample 8, followed by only a very slight decrease between samples 8 and 9. It is extremely improbable that any combination of self-compensating errors should result in such uniformity of silicon-content. This constancy in the samples 6, 7 and 8, all taken after the iron had melted, agrees with the constancy from the samples

1 to 5, when some of the iron was still melting, so that the removal of the silicon seems to have been extremely slow during the relatively large decarburization of the first-melting part of the iron, during the slighter decarburization of the later-



A. Charge not fully melted at throat; metal and slag sluggish. *B.* Not yet all melted at throat; almost no boiling. *C.* Nearly all melted. *D.* Nearly all melted; a little mushy. Slag now skimmed. *E.* New slag formed spontaneously.

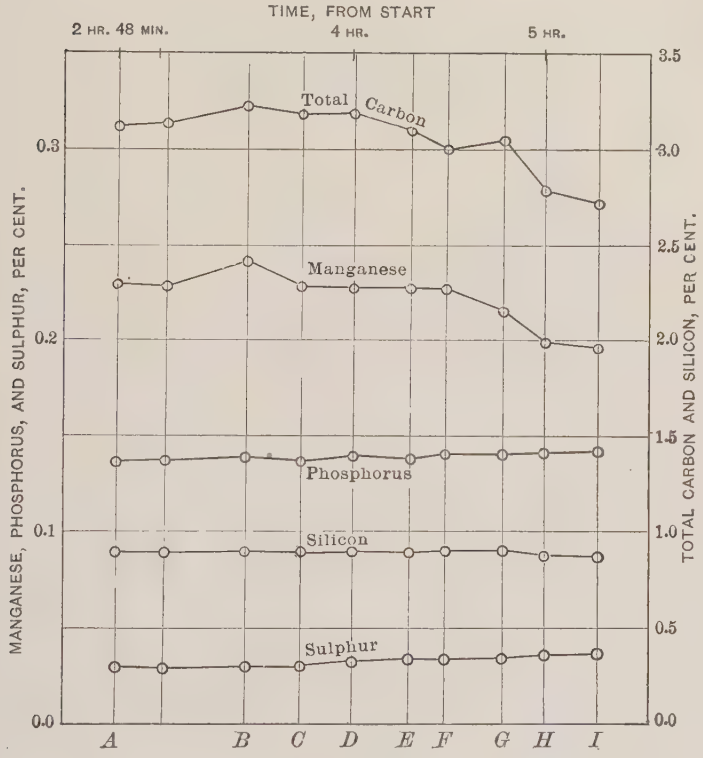
FIG. 1.—COMPOSITION OF SLAG IN AIR-FURNACE PROCESS.

melting iron, and while the whole lay fully molten on the hearth.

Why does the silicon-content remain so constant? Partly, at least, because it is so small relatively to the carbon-content.

Such constancy is certainly extremely rare in the acid Bessemer and acid open-hearth processes; but where the silicon-content decreases very slowly in these, there it is, in general, very small.

To throw light on this point, we have condensed in Table II.



A. Charge not fully melted at throat; metal and slag sluggish. *B.* Not yet all melted at throat; almost no boiling. *C.* Nearly all melted. *D.* Nearly all melted; a little mushy. Slag now skimmed. *E.* Pours thin. *F.* First of melter's sample. *G.* Last of melter's sample; new slag formed spontaneously. *H.* Begin pouring 5 min. after this. *I.* Charge nearly finished, as indicated by much sparkling; 5 min. later, all out.

FIG. 2.—COMPOSITION OF IRON IN AIR-FURNACE PROCESS.

data from all the investigations of which we know, that show the progress of the removal of the several non-ferrous elements in the acid open-hearth process. For sake of comparison, case No. 1 gives data concerning an acid Bessemer heat, in which the removal of silicon was decidedly less than in any other of which we know. In this case the initial silicon-con-

TABLE II.—*The Removal of Silicon as Affected by the Silicon-*

Kind of Process.	Number of Case.	Authority and Reference.
Acid Bessemer.	1	Müller, *Stahl und Eisen*, vol. x., No. 2, pp. 115–9 (Feb., 1890).
	2	Campbell, *Trans.*, xix., 160, Group I. (1890–1).
	3	*Iron Age*, vol. liv., No. 25, p. 1113 (Dec. 20, 1894).
	4	Campbell, *Trans.*, xxii., 416, Group II. (1893).
	5	Wahlberg, *Journal of the Iron and Steel Institute*, vol. lxi., (1902, No. I.), p. 340, Table II.
	6	Odelstjerna, *Journal of the Iron and Steel Institute*, vol. xxix. (1886, No. II.), p. 938, Charge No. I.
	7	Campbell, *Trans.*, xix., 160, Group II. (1890–1).
	8	*Idem*, xxii., 416, Group I. (1893).
	9	*Idem*, Group IV.
	10	*Idem*, Group III.
	11	Odelstjerna, *Journal of the Iron and Steel Institute*, vol. xxix. (1886, No. II.), p. 938, Charge No. II.
	12	Campbell, *Trans.*, xxii., 416, Group V. (1893).
	13	Thwaite, *Proceedings of the South Staffordshire Institute of Iron and Steel Works' Managers*, vol. vi. (session 1890–91), diagram facing p. 38.
Acid Open-Hearth.	14	Wahlberg, *Journal of the Iron and Steel Institute*, vol. lxi. (1902, No. I.), p. 340, Table I.
	15	Richards, *Journal of the Iron and Steel Institute*, vol. lxi. (1902, No. I.), p. 64.
	16	Odelstjerna, *Journal of the Iron and Steel Institute*, vol. xxix. (1886, No. II.), p. 938, Charge No. III.
	17	Cubillo, *Journal of the Iron and Steel Institute*, vol. lxiii. (1903, No. I.), p. 280.
	18	From author's private notes.
	19	Thwaite, *Proceedings of the South Staffordshire Institute of Iron and Steel Works' Managers*, vol. vi. (session 1890–91), diagram (Steel Co. of Scotland practice) facing p. 38.
	20	*Idem*, Diagram (Barrow Steel Works).
	21	McWilliam and Hatfield, *Journal of the Iron and Steel Institute*, vol. lxi. (1902, No. I.), p. 58.
	22	*Idem*, p. 59.
	23	*Iron Age*, vol. liv., No. 25, p. 1112 (Dec. 20, 1894).
	24	Mehrtens, *Stahl und Eisen*, vol. xi., No. 9, p. 709 (Sept., 1891).
	25	Addie, *Engineering*, vol. lii., pp. 103–4 (July 24, 1891); *Journal of the Iron and Steel Institute*, vol. xl. (1891, No. II.), pp. 266–7.
	26	Kollmann, *Verhandlungen des Vereins zur Beförderung des Gewerbfleisses*, 1880, p. 221; Ledebur, *Handbuch des Eisenhüttenkunde*, 2d ed., p. 961 (1894).
	27	Willis, *Journal of the Iron and Steel Institute*, vol. xvi. (1880, No. I.), p. 90, Charge II.
	28	*Idem*, Charge I.
Air-furnace.	29	Air-furnace, present paper.

Content, and by the Ratio of the Carbon-Content to the Silicon-Content.

Number of Case.	Silicon.			Carbon.		Initial Ratio C / Si	Loss of C ÷ Loss of Si.	Ore Added or Not.
	Quantity in Initial or First Sample Recorded.	Next Sample.		Quantity in Initial or First Sample Recorded.	Loss of C at Time of Next Sample.			
		Time After Initial (or After First Sample).	Loss of Si in Per Cent.					
	Per Ct.		Per Ct.	Per Ct.	Per Ct.			
1	0.70	3 min.	0.14	3.65	1.15	5.21	8.2	.....
2	0.02	.....	0.00	0.54	0.41	27.0	∝	Yes
3	0.09	.....[a]	0.05	2.19	2.13	24.3	42.6	Yes
4	0.01	.....	0.00	0.18	0.10	18.0	∝	Yes
5	0.121	1 hr.	0.088	2.00	0.25	16.5	2.8	Yes
6	0.28	1 hr. 30 min.	0.01	3.6	0.26	12.9	26.0	No
7	0.05	.....	0.04	0.64	0.52	12.8	13.0	Yes
8	0.03	.....	0.01	0.36	0.28	12.0	28.0	No
9	0.07	.....	0.05	0.61	0.53	8.7	10.6	Yes
10	0.04	.....	0.01	0.32	0.24	8.0	24.0	Yes
11	0.54	55 min.	0.01	3.6	0.2	6.7	20.0	No
12	0.09	.....	0.07	0.57	0.49	6.3	7.0	Yes
13	0.50	1 hr.	0.25	1.9	0.1	3.8	0.4	.....
14	0.611	1 hr.	0.242	2.20	Gain 0.10	3.6	.....	Yes
15	0.70	20 min.	0.24	2.15[c]	0.31	3.1	1.3	Yes
16	1.19	1 hr. 25 min.	0.40	3.70	0.08	3.1	0.2	No
17	0.59	30 min.	0.00	1.66	0.06	2.8	∝	No
18	1.38	....	1.36	3.81	0.81	2.8	0.60	No
19	1.48	1 hr.	0.83	3.21	0.33	2.2	0.40	....
20	1.10	30 min.	0.36	2.46	0.61	2.2	1.7	.....
21	0.78	35 min.	0.43	1.60[c]	0.06	2.1	0.14	Yes
22	0.80	40 min.	0.52	1.60[c]	0.02	2.0	0.04	Yes
23	0.756	30 min.	0.263	1.38	0.00	1.8	.....	No
24	0.77	7 hr.	0.42	1.30	0.50	1.7	1.2	No
25	1.350	.....	1.154	1.755	0.1	1.3	0.09	Yes
26	0.480	7 h r.	0.330	0.497	0.437	1.04	1.32	No
27	1.60	30 min.	0.690	1.34	0.00	0.84	.....	.....
28	1.281	30 min.	0.163	1.00	0.00	0.78	.....	.....
29	0.89	2 hr.[b]	0.00	3.10	Gain 0.02	3.5	.....	No

[a] The initial sample was taken after the first addition of ore, and the "next sample" at tapping time.

[b] The silicon-content remained constant for two hours.

[c] Reported as combined-carbon.

tent was unusually low, and the initial temperature unusually high.

In every case in Table II. in which the removal of silicon between two successive samples is either zero or very small, the silicon-content itself is very small, either absolutely or relatively to the carbon-content. To the former class belong cases 2 to 5 inclusive, 7 to 10 inclusive, and 12, in none of which is the initial silicon-content greater than 0.121 per cent. To the latter class belong cases 6, 11, and 17, which have respectively 12.9, 6.7, and 2.8 times as much carbon as silicon to start with.

Of course, the silicon-content and its relation to the carbon-content do not alone determine the order of removal of silicon and carbon. For instance, in cases 14 and 15, though there is but little silicon present, 0.60 per cent. in one case and 0.70 per cent. in the other, and though there is more than thrice as much carbon present as silicon, yet the removal of silicon is very considerable. It is well known that a low temperature favors the oxidation of the silicon rather than of the carbon, and so does basicity of slag. The influence of basicity is shown clearly when we pass to the extreme case, that of the basic open-hearth process, in which the silicon is removed very rapidly. In case 15, though the slag nominally contained 46 per cent. of silica at the time of the first sample, and thus was but slightly more basic than usual, yet the large addition of ore then made, 1 ton, had the double effect, first of lowering the temperature and thus favoring the removal of silicon, and then of making the slag effectively basic at the places where the actual oxidation is most rapid, *i.e.*, in the immediate neighborhood of the individual lumps of ore; and it is the basicity in these parts of active oxidation that counts in determining the relative oxidation of carbon and silicon. Hence the large removal of silicon in spite of the low silicon-content and high ratio of carbon to silicon.

But in case of the air-furnace operation which we here report, the high ratio of carbon to silicon, 3.5 : 1, is probably the condition which retarded the removal of silicon, because the low temperature should have favored the removal of silicon, and the acidity of the slag is not great enough to explain the extreme slowness of the removal of silicon. That the temperature was very low at the beginning of the process is shown by

the fact that the bath was but imperfectly molten, in spite of the fusibility implied by its having more than 3 per cent. of carbon. Neither the silica-content of the slag, from 46 to 62 per cent., nor its joint silica- and alumina-content, from 50 to 67 per cent., is unusually great.

The presence of silicon favors graphitization, *i.e.*, the change of the carbon from the combined to the graphitic state, certainly during solidification and probably in the annealing of the malleablizing process. As far as the annealing of the castings itself is concerned, therefore, the silicon-content should be high, in order that, by favoring graphitization, it may either shorten the time needed for the annealing, which of course consists essentially in graphitization, or make the annealing more thorough. But the permissible silicon-content is limited by the need of having the "hard metal" itself free from initial graphite formed during and shortly after solidification (primary and eutectic graphite), the coarse flakes of which would weaken the iron by breaking up its continuity. This initial graphite, of course, persists through the annealing; and indeed, even if it could be removed by oxidation, the cavities where it existed would remain, and would weaken the mass about as much as if the graphite itself were still there.

Hence the practice of using much steel scrap in order (1) to reduce the initial carbon, (2) thereby to lessen the tendency to the formation of initial graphite, *i.e.*, graphitization during and immediately after solidification, (3) thereby in turn to permit a greater silicon-content without causing initial graphite to form, and (4) thus to shorten the annealing period or to make the annealing more complete.

Appendix.

Description of the operation:

The charge consisted of: steel scrap, 500; malleable cast-iron scrap, 500; hard metal or sprue, 7,200; pig-iron, 10,000; total, 18,200 pounds.

This was charged at 7.30 a.m., May 20, 1905, in an air-furnace of the usual type, a reverberatory furnace burning bituminous (Westmoreland) coal on a common grate.

Sample 1, at 10.18 a.m. The charge was not fully melted at the throat of the furnace, and the metal and slag were sluggish.

Sample 2, at 10.33 a.m.

Sample 3, at 10.58 a.m. Not yet all melted at throat. Rabbled the bath thoroughly before sampling. About 2 in. of thick, sluggish slag on the bath.

Sample 4, at 11.15 a.m. Nearly all melted. Did not rabble before taking this sample.

Sample 5, at 11.31 a.m. Rabbled very thoroughly before sampling. The metal was a little mushy when poured into the test-mold. Immediately after this sample the whole of the slag was skimmed off. It was mushy.

Sample 6, at 11.49 a.m. Taken immediately after rabbling. Was very fluid.

Sample 7, at 12 m. Taken just after rabbling. This was the first of the "melter's samples."

Sample 8, at 12.18 p.m. Stirred before sampling. No boiling in the furnace. The sample sparkled slightly. This was the last of the "melter's samples."

Sample 9, at 12.31 p.m. Stirred thoroughly before sampling.

12.36 p.m. Began pouring. Metal sparkled much in ladles, but did not boil.

Sample 10, at 12.47 p.m. Sparkling very much.

12.52 p.m. The iron has all run out.

"Hard metal" means the white cast-iron as prepared in this air-furnace process, preparatory to its conversion into malleable cast-iron by the graphitization of its carbon in the annealing process.

The Professional Examination of Undeveloped Mineral Properties.

BY CHARLES CATLETT, STAUNTON, VA.

(Chattanooga Meeting, October, 1908.)

THE terms "developed" and "undeveloped" are necessarily relative and cover a wide range; but the latter is here applied to cases in which the information at hand falls short of a clear demonstration of values, and conclusions must be reached from a study of the geological conditions of the particular deposit, interpreted in the light of similar and closely-related deposits. Such an inquiry might be characterized as an effort to reach reliable conclusions from insufficient data. But the engineer

engaged in commercial work finds himself often face to face with apparently such problems; and while he may realize with regret that a small amount of intelligent development would add enormously to the value of his conclusions, he is compelled to recognize the demand of a business situation for a prompt decision as the basis of immediate action.

In a case within my knowledge, the question involved was the purchase of 50,000 acres of coal-land, and 10 days was the limit within which this question must be decided. Of course, it was impossible to make a satisfactory examination within that time; but the instructions were "to make the most complete examination and the most positive recommendation possible." Such instructions are often given; for, no matter how much time an investor may have, he is likely to put off his decision until the last moment, and then discover that he needs more light and advice. It does not follow that engineers, after full explanation and mutual understanding of the situation, should decline to undertake examinations thus limited. In the case just mentioned, while only the broadest view was possible, half the time available was sufficient to show that the property was a good purchase. The timber alone was subsequently resold for more than the entire purchase-price of the land. In other words, it was easy to determine the advisability of the bargain offered, though impossible to fix definitely, in so limited a period, the value of the tract.

Of the properties thus offered for sale, a large proportion can be promptly condemned as obviously not worth the price asked; a much smaller proportion can be quickly pronounced good from the immediate standpoint of the investigation; and there remains another small proportion, with regard to which the pros and cons are hopelessly mixed, and no decisive opinion can be safely pronounced without further evidence.

It is, however, rare that a deposit is so irregular that valuable conclusions may not be reached by a careful and detailed examination of such facts as nature has already exposed. But it is equally rare that a small amount of intelligent prospecting fails to give valuable information, especially if directed so as to disclose the local geological conditions, and not simply to expose a measurable quantity of ore. Such preliminary work should always be done, if possible.

I have known a leading dealer in mineral properties to object to geological examinations, because, as he said, the investigator always wanted some prospecting done, and if prospecting was done he could see for himself—forgetting that a true interpretation of the facts presented was possible only to those familiar with the language which nature employed in that particular instance. A striking instance of the advantage of broad scientific knowledge is seen in the description by Thomas F. Walsh, of his discovery of the Camp Bird lode. A rock attracted his attention "as having a strong indication of gold in or near it." The men who had previously overlooked this indication "were no ordinary prospectors;" yet they had failed to see the real values thus disclosed. Unfortunately, the skilled observer is not always finding values; but rather the reverse. He usually finds sound reasons for condemning where the less informed are enthusiastic.

Another example of the need of scientific knowledge is presented when the geological formation to which a mineral deposit belongs is a sufficient factor in the determination of its value. Practically, of course, the geological age of a deposit may mean nothing, if it has been so far developed as to show a satisfactory amount and quality of ore. But in arriving at an opinion before such development, the identification of the geological horizon may be important. Thus, the fossil iron-ore of the Clinton formation is usually so uniform in its superficial characteristics as to be identified by the naked eye. The recognition of these particular beds in a new locality would warrant a reasonable expectation that they would show a development similar to the same beds elsewhere. But if a bed of somewhat similar appearance were shown not to belong to the Clinton formation, we would know that its laws as to occurrence and extension needed to be investigated as a new proposition. Again, if the great Pocahontas coal-seam could be identified in territory where it is not known to exist, far more favorable expectations might be entertained from such a discovery than from an equal number of openings on a new and unknown seam. I know of a case in which a net loss of several hundred thousand dollars will be incurred from a failure to identify properly the geological measures to which a given deposit belonged. The positive, clear, valuable and available

facts are usually few in comparison with the size and extent of an undeveloped property, and it is peculiarly important that they should be seen, studied, and comprehended in true perspective.

This raises the question whether an examiner should see previous reports before his own examination. So far as a statement of facts is made he should certainly do so, in order that nothing may be overlooked, and that certain observations may be verified; but it is less important to know the opinion of others in advance of forming one's own conclusions. Especially is it necessary to see things; and the man who from physical inability or laziness is not capable of making unusual exertion to cover ground, and to see everything that is to be seen, is no longer capable of doing good work in the examination of undeveloped properties, whatever may be his mental acquirements.

The accessible facts must be studied in as great detail as the conditions of the examination permit. Nothing should be taken for granted that can be proved; nothing guessed at that can be measured or weighed; and nothing left to another that can possibly be done by the investigator himself.

While it is always desirable, and at times necessary, to study the facts in the utmost detail, it cannot be denied that practically-accurate conclusions may often be reached from what appears to be a superficial examination. Practice develops an ability to recognize and give due weight to important facts, and to ignore those immaterial to the purpose of the particular examination, which enables a clear and definite impression to be reached with great rapidity and with unconscious effort. Nine times out of ten this first impression is correct; but it should always be fortified, if possible, with more detailed investigation.

But a mere examination of a property with an exact statement of what is seen is as colorless as a flash-light photograph. It is information, but not advice. Beyond the questions of quantity and quality, and the cost with which a mineral can be won, are those of the present and future market, the present and future labor-supply and transportation, and the peculiar conditions affecting the operations of one's client. These require a much broader study in the light of business experience,

and furnish the reason why the conclusions of many men of eminent scientific attainments have been disappointing commercially. In particular must the investigator put himself in the position of his client, and endeavor to decide what is best for him, and while presenting the facts in such a way that his client may reach his own conclusion, the investigator cannot hesitate to give his own honest and unbiased advice. It is not often that a client knows exactly what he wants, beyond the fact that he is seeking profit, and is disappointed if the investigation fails to promise it.

In my experience, the most clear and definite instruction was received in a commission which is interesting as illustrating one of the periods when the trend of the development of the iron-industry was uncertain. In 1895 the Mesabi range was still something of an experiment, and Andrew Carnegie's attention was called to the fact that the furnaces in Virginia had been able to peg along, in good times and bad, and had shipped basic iron into Pittsburg when a large proportion of the furnaces there were cold on account of low market-prices. If those conditions could be maintained on a sufficient scale, why should he not acquire the control of that ore-region, and establish a direct railroad connection from Pittsburg?

If the Virginia field would furnish a certain tonnage of ore, of a certain quality, particularly in regard to phosphorus, which could be mined at a certain average cost (and all these requirements were reasonable), the problem was an easy one. Having been engaged to investigate these questions, I got control, in my own name, of several hundred thousand acres of land, which were prospected in a quiet way, the analytical work being done in my own laboratory, and I also investigated the general conditions surrounding the resources of the operating companies. Without reflecting on the many valuable individual properties in that region, the investigation showed conclusively that the field as a whole could not be expected to meet the requirements of Mr. Carnegie; and he turned more energetically to the Northwest. Had I been able to reach a different conclusion, the industrial history of the States would have been changed.

During this time, Mr. Carnegie's attention was called also to the Tennessee Coal, Iron & Railway Co. He said it was true

that the South fixed the price of foundry-iron, and was foolishly giving away its patrimony by selling it so cheaply; but that he could not have his name connected with that "football of speculation." Nevertheless, I think there would have been a fair margin of profit between the price at which the T. C. & I. could have been bought then, and the price at which it could be bought now.

Both of these decisions were characteristic of that wonderful organization which Mr. Carnegie had built up, which knew what it wanted, and was willing to pay liberally to secure it, but when the results of inquiry were unsatisfactory promptly dropped the proposition, considering its money well spent in securing accurate information, even though that information was disappointing. This is not usually the case.

Clients who call for opinions involving enormous sums often fail to explain the true purposes of the inquiry, and pay with reluctance the cost of reliable information, if it is unpleasant to them. It is not strange, therefore, that expert reports are often scattering and indefinite. No examination can be satisfactory if the full plan of the principals is not known to the investigator. Every bit of information he may collect is of greater value if he collects it with full knowledge of the use to which it is to be put. If he is not worthy of such full confidence, his advice in an important matter should not be asked.

Almost as difficult a proposition, is one in which the instructions are limited because the client himself lacks the necessary knowledge. He has been told of a profitable opportunity, but has no very clear idea concerning it. In one instance, I received from a client, in a matter involving an investment of five or six million dollars, what he regarded as full instructions. But these instructions simply meant, "Would this transaction be a profitable one?" Before the mineral property which was the basis of the proposition could be intelligently investigated, it was necessary to find out what the client actually wanted and what he could do with the property if he bought it. In another very important case, the only instructions were, "We have bought a property, and we want to know what we have." It was left to the investigator to decide what facts would be of most value, and in what form those facts were to be presented.

The complete confidence which should be given to the inves-

tigator requires, on his part, that this confidence should be kept inviolate, with regard both to his client's plans and to the property examined. The examination of an undeveloped property frequently implies plans for railroad-extension and development, and is rarely heralded with a trumpet. It is a very desirable accomplishment to be able to make a satisfactory investigation without calling undue attention to it by an exaggerated air of mystery, and without giving a suggestion either of the name of the client, the purpose of the investigation, or the conclusions reached.

This caution is equally obligatory with regard to an opinion formed before engagement for the particular examination in question, and upon evidence not acquired in the course of that work. In other words, the acceptance of the professional duty of examining a given property, and making a confidential report concerning it, involves a tacit obligation to say nothing whatever about it thereafter, which could affect the client's interest. What the adviser knows or thinks he knows about it already, belongs to his client, as well as what he may find out afterwards. Any intimation of his views, doubts, or impressions, whether favorable or unfavorable in tendency, may seriously interfere with his client's plans. Moreover, such an intimation, if unfavorable, will certainly offend the owner of the property, and perhaps precipitate an attack upon his own ability or motives. Such criticism he must, of course, be prepared to endure if his report should be published. But an unfavorable report, defeating a proposed purchase, is not likely to be published; it is no part of an expert's business to put unnecessarily upon public record a discouraging estimate of the value of the property of a party with whom he has had no professional relations; and it is unwise, as well as improper, to volunteer any statement of that kind. Finally, the data of fact or opinion contained in a professional report should not be published by the author without the consent of his client. The lapse of a long time may constitute a probably sufficient ground for believing that such a publication is permissible; but the client's explicit consent is a much better ground, and can be, in most cases, easily secured. If it is refused, the refusal should be conclusive.

The question, whether a man should undertake the exami-

nation of a mineral property, when he believes in advance that his report will necessarily be unfavorable, answers itself. He is bound, of course, to advise his client of his pre-existing unfavorable opinion, and the grounds upon which it is based. After that, he can do whatever he is asked to do, in the way of a detailed examination of the facts, and a revision of his former judgment, or the foundation of an independent new one, upon evidence personally studied, and perhaps previously unknown.

The influences tending to affect an expert in one or the other direction are, perhaps, equally balanced. When personal friends are interested in a favorable result, the wiser course, probably, would be to decline the offered engagement. But in many other instances, also, it is much pleasanter to gratify both buyer and seller by an optimistic interpretation of the facts. On the other hand, the reputation of being a severe judge, and of condemning a proposition of purchase unless it offers more than ordinary safety and profit to the purchaser, is often professionally valuable to one whose employers are looking only for exceptionally good bargains; and perhaps his strongest and most subtle temptation may be that of protecting himself and enhancing the value of his favorable reports by doing less than justice to the property examined. This influence should not be permitted to prevent the frank and fair expression of opinion as to the case in hand.

The form of a report is determined primarily by its purpose. A scientific treatise on geologic conditions and the formation of ore-deposits may be without immediate and pertinent value, however lavish and profound. I have handed such a report to a client, only to have him ask, as he tossed it aside, "Shall I buy the property, or not?" Nevertheless, reports have a habit of living long past their allotted time. They are read by people for whom they were never intended, and under circumstances under which they were never expected to appear; and they should, if possible, carry on their face such evidence as will show that they were based upon thorough investigation and intelligent interpretation of the facts. Who has not seen some one of his reports arise like Banquo's ghost, when he least expected it, and regretted that he had not, in that particular case, put his views in better shape? I have an example before me at present. A report of mine, written *en route* in great haste,

has turned up in an important lawsuit. The conclusions and the advice were sound, but I heartily wish the original, which properly met the conditions at the time, had been put in more attractive and complete form.

From some standpoints, a report cannot be too elaborate; but besides preparing it in that way for the benefit of those who may come after, and who may want to know the opportunity enjoyed for reaching a correct conclusion, it should be digested and stated in plain, simple, straightforward style, and in the shortest possible space, so that the client may know what is the summary of the investigator's conclusions and advice. A busy man often will not wait for a careful report; and really does not want it.

In one instance, involving the question of a bond-issue of about $800,000, I had planned to make an exhaustive investigation and report. At the end of a very short examination my principal asked what statement he was in a position to make, and was advised that I was willing to say that there were so many tons of ore of a certain grade, and that I hoped to be able to increase that estimate largely by additional investigation. His reply was, "If you can write me a short letter stating that fact, it is exactly what I want, and I would rather not have any other report." Subsequent drilling disclosed about two and a half times this minimum.

I have reason to think that one of the most satisfactory reports with which I have had anything to do was confined to a tabulated statement on one page, though months were afterwards consumed in elaborating the conclusions there submitted.

I venture to say, therefore, that a satisfactory report on an undeveloped mineral property should present, in the shortest possible space, and in the clearest, most positive and definite language, the conclusions of the investigator; and also, if the conditions permit, should set forth, in their proper order, all those facts of geography, topography, geology, etc., which would enable one of fair intelligence, and to whom the subject is new, to understand it thoroughly, and to form an individual opinion as to the care with which the investigation had been made and the propriety of the conclusion reached; bearing always in mind the point from which the examination was approached, and also that every one who examines a mineral property stands in a more or less fiduciary relation to both the buyer and the seller, and that he owes a duty to each.

The South African Tin-Deposits.

BY WILLIAM R. RUMBOLD, ORURO, BOLIVIA, SOUTH AMERICA.

(New York Meeting, April, 1907.*)

When I was in South Africa during the latter part of 1904, there were three known tin-fields, which may be called the Cape Town, the Bushveld, and the Swaziland fields.

The Cape Town Tin-Field.

This district is situated near Kuils river, 20 miles west of Cape Town, and 5 miles from Kuils River station on the Cape Town & Stellenbosch railway. The deposits are in granite, intrusive in the Malmesbury beds, which are supposed to be of Silurian or Cambrian age. The granite, resisting denudation better than the surrounding shales, has formed low-lying hills, the highest of which, the Bottlelary Berg, is 1,510 ft. above sea-level. The rock consists of quartz, orthoclase feldspar, and biotite mica; tourmaline is also present. The feldspar occurs in large crystals. The rock is generally decomposed on the surface, and the decomposition sometimes extends as deep as 100 ft. This formation—granite intrusive in Silurian shales—extends for many miles to the northeast, but so far as I know the Kuils River deposits are the only ones now known to occur in it, though probably further prospecting will reveal other deposits. This particular discovery was accidental; alluvial tin being found in the course of boring for oil.

The Kuils River Deposit.

This deposit comprises both lode and alluvial tin. The lode strikes N–S., and dips about 30° E. into the hill. It is 5 ft. wide, and composed of dense white quartz, in which pinkish-gray cassiterite is distributed in large crystals. The vein also contains considerable quantities of wolframite.

Fig. 1 is a rough plan of the relative positions of the lode and the alluvial deposit, the latter, as shown, following the course of a stream, while the lode crops out on the hillside

* Revised proof received from the author too late for incorporation in Vol. XXXVIII., to which it chronologically belongs.

above. The bed averages 2 ft. in thickness, rests on a bed-rock of decomposed granite, and contains boulders of white quartz up to 2 ft. in diameter, and stones of the same material. It carries also between 15 and 20 lb. of cassiterite to the cubic yard. The cassiterite in the alluvium has the same pinkish-gray color as that in the lode above, from which it was obviously derived.

The origin of the lode itself is not so evident. At first sight,

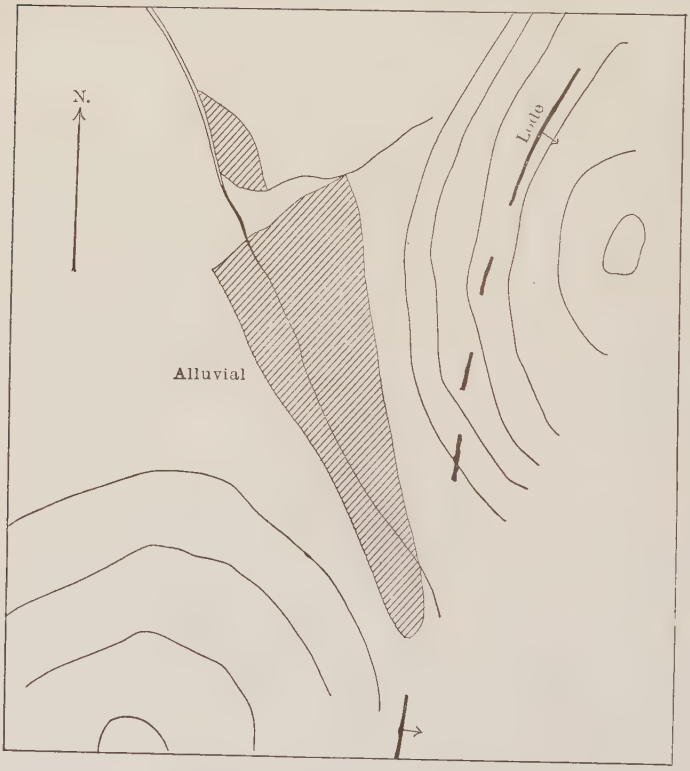


FIG. 1.—PLAN SHOWING THE KUILS RIVER TIN-DEPOSIT.

the large isolated cassiterite crystals reminded me of a pegmatite-vein; but feldspar was not visible to the naked eye. There was a certain amount of chlorite, but no muscovite, nor did the vein in any way resemble a greisenized crack in the granite.

It might be either a fissure in the granite, in which meteoric waters had deposited quartz and mineral, or it might be an instance of magmatic segregation accompanied by pneumatolytic action. I incline to the latter hypothesis.

The Bushveld Tin-Field.

This district is situated on a farm called Enkeldoorn, 35 miles NE. from Hatherley, on the line between Pretoria and Delagoa bay. The country is granite; I did not examine its boundaries, but it is probably extensive. It is composed of quartz and feldspar, with very little mica. The feldspar is orthoclase, usually pink, and sometimes occurring in large well-formed crystals. The grain of the rock varies. Occasionally (and generally in the vicinity of the lodes) it is fine-grained, and approaches an aplite.

The ore-body, as I saw it, had a N–S. strike, dipped 80° W., was 1 ft. wide on the surface, and opened to 4 ft. in width a little below. It had no well-defined walls. The gangue was quartz, decomposed granite, iron oxide, and a little chlorite; the amount of cassiterite did not appear to be large.

The deposit had only recently been discovered, and very little work had been done; in fact, none of the trenches had passed through the weathered zone, and no unaltered rock was to be seen. Owing to the shortness of my visit, and the small amount of work done, I hardly dare pass an opinion as to the origin of the deposit, but, on general principles, it should prove no exception to other occurrences of tin-ore in granite.

There was considerable excitement in Johannesburg over the new discovery at that time; and, according to newspaper reports, there have been other discoveries and a "boom" in tin-shares since. As to these new discoveries, I know nothing; but, if they resemble the one I visited, I fear they have been rated more highly than they will be when examined coolly and carefully by engineers familiar with tin-deposits.

The Swaziland Tin-Field.

This district, situated on the border of the Transvaal and Swaziland, extends from Oshoek to Steynsdorp, and southeast to Embabane. The chief geological features are a hornblende granite, and a belt of highly metamorphosed, contorted schists, which divide the granite into two sections, the granite being intrusive in the schists. Fig. 2 is an ideal section from Oshoek to Embabane.

The granite mass varies in texture in different parts of the district, but at Oshoek it is fine-grained. Both the granite and

the schists are traversed by numerous dikes called diabase, but seen under the microscope, they prove to be diorite; moreover —a much more important matter for the tin-miner—the rocks are seamed by pegmatite-veins, which are often so close together as to form a stockwork. These pegmatites are the source of the tin.

The Oshoek Deposits.

These deposits consist of stanniferous pegmatites and the alluvial beds formed from them. The granite is of the type mentioned above, but frequently it is foliated, and develops a banded structure, so as to be almost a gneiss. The pegmatite-veins exhibit a coarse, crystalline intergrowth of quartz and feldspar, carrying a little hornblende, and sometimes a few large crystals of muscovite. Garnet, monazite, euxenite, and

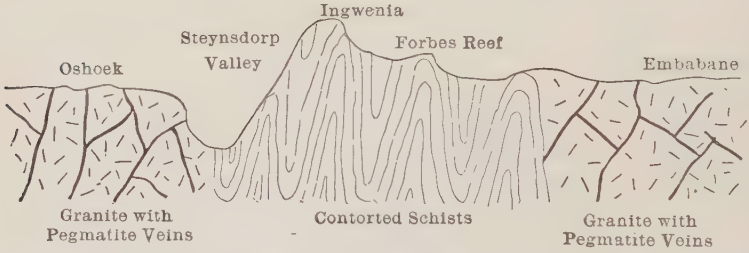


FIG. 2.—IDEAL SECTION OF THE SWAZILAND TIN-FIELD.

æschynite are found with the cassiterite, which occurs usually in large isolated crystals. The veins are in belts separated by tracts of barren granite.

Fig. 3, a sketch of the sections shown in the side of a cut at the mouth of a tunnel near Oshoek, shows the pegmatite-veins in the hornblende granite.

Fig. 4, a sketch from the wall of a tunnel near Oshoek, shows, in addition, a dike of diorite.

The proportion of cassiterite in the pegmatite-veins is small: 0.25 per cent. of "black" tin was the yield of the best sample I saw; and that was taken from the decomposed outcrop, where the tin had been, to a certain extent, concentrated. The veins, which may vary from 20 ft. to a few inches in width, are unfortunately too far apart to permit cheap extraction by quarrying.

The alluvium, which occurs in patches, along the streams in

the valleys, is not of great importance, although it has been the principal source of the tin thus far produced.

The Forbes Reef Deposits.

These deposits are situated 11 miles NE. of Oshoek, on the eastern side of the Ingwenia range. They occur in the belt of schists which divides the granite mass. These schists

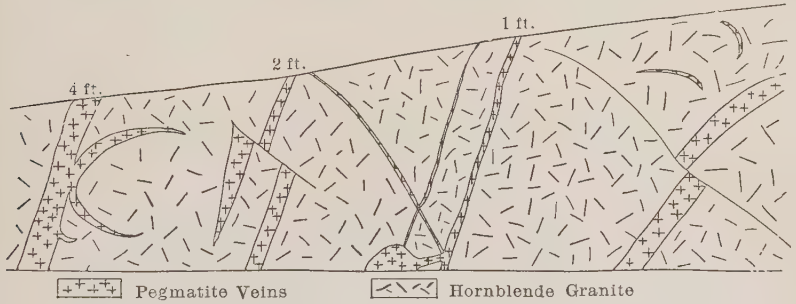


Fig. 3.—Pegmatite-Veins in Hornblende Granite, near Oshoek.

vary greatly; but they consist chiefly of talc, hornblende, and actinolite; and each variety is found repeating itself as one travels east or west, across their N–S strike. They carry, besides cassiterite, a great number of minerals, including gold,

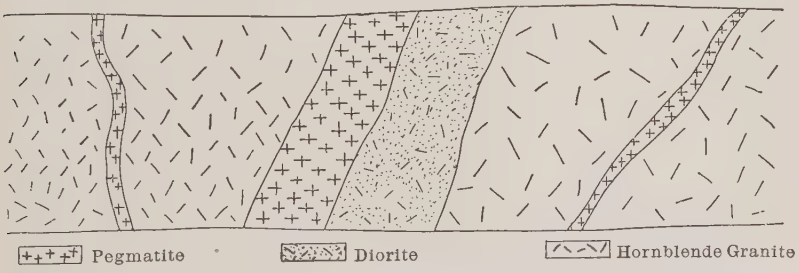


Fig. 4.—Pegmatite-Veins and Diorite-Dike in Hornblende Granite, near Oshoek.

cinnabar, scheelite, bismuth, monazite, euxenite, æschynite, copper and iron pyrites, pyrrhotite, margarite, tourmaline, hornblende, talc, and actinolite. The tin occurs in veins in the schists themselves, and also in a fine-grained dike of aplite, in which the quartz and feldspar are intimately mixed, and which, having the same strike as the schists, intersects them by virtue of its much flatter dip.

Fig. 5, a section near Forbes reef, shows both forms of occurrence. In the dike, the tin-ore is distributed throughout, in crystals and pockets; but not much could be learned about the deposit at the time of my visit, since only a few prospecting-trenches had been dug. The occurrence in the schists is most remarkable. The veins are not more than an inch wide, dip at a steeper angle than the aplite, and carry flat, tabular crystals of cassiterite imbedded in clay. The crystals have a fine amber color, and are laminated and brittle; sometimes, when scratched, they will break into powder like a Rupert's drop, showing the intense strain and pressure to which they have been subjected. Margarite is found in the schists in bunches, sometimes carrying a little tin.

The modern theory of pegmatite-veins is that "while they may be to some extent the result of consolidation from a

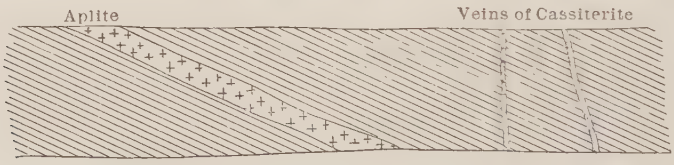


Fig. 5.—Aplite-Dike and Cassiterite-Veins in Belt of Schists, near Forbes Reef.

molten state, they are very largely of pneumatolytic origin" . . . . . that they "are formed *after* the consolidation of the main mass of the igneous rock, and are to be considered as the last results of magmatic differentiation."[1] There seems no reason to depart from this theory for the origin of the pegmatite-veins of the Swaziland tin-field, and one can only suppose the aplite-dike and veins of Forbes reef to be likewise due to the pneumatolytic after-action of the underlying granite.

General Conclusions.

From an economic point of view, the South African tin-fields are, so far, distinctly disappointing. The Cape Town field offers the most encouraging prospect for future development. With regard to the sanguine views so early expressed concerning the

[1] "The Character and Genesis of Certain Contact-Deposits," by Waldemar Lindgren, *Trans.*, xxxi., 242; reprinted in *Genesis of Ore-Deposits*, 2d ed., p. 732 (1902).

Bushveld and Swaziland fields, one cannot help suspecting that some of the Johannesburg experts, though they may know all that is worth knowing about gold, do not possess a corresponding knowledge of tin-deposits and what constitutes payable and unpayable tin-ore. This is not necessarily to their discredit; an expert on one subject is not, for that reason, an expert on another.

I wish to repeat that the opinions here expressed are based upon the evidence as it existed a little less than two years ago, and not upon developments since made. On the other hand, many of the exaggerated valuations and over-confident prophecies to which I have referred must have been based upon nothing more than I had observed; for they were published at or very shortly after the time of my visit.

To take, for instance, the pegmatite-veins: it was then doubtful whether the ore from these would average 0.25 per cent. of "black tin." When metallic tin is worth £130 a ton, the value of ore containing 0.25 per cent. of "black tin" is 4s. per ton—a pretty low value for ore that has to be mined and not quarried, where water-power is not unlimited, and when the place is 80 miles from a railway.

Nor does the character of the pegmatite-veins warrant the confident expectation of richer ore in depth. I was informed, for instance, that the juncture of the pegmatite and the "diabase" would probably have the effect of enriching the deposit. But this "diabase" is distinctly later than the pegmatite, and is never, or very rarely, a tin-bearer; consequently, the idea that it will enrich a poor deposit is highly improbable.

Moreover, the method of sampling, which on the Rand has been reduced to an almost exact science, seems to be a matter of mere specimen-picking, when a tin-lode is the subject instead of a banket-reef; whereas, the tin-lode should have double care bestowed on it, because its values are bound to be more erratic than those of the Rand formation.

I sincerely wish for the South African tin-fields all the success new mining enterprises deserve, but before they begin to figure in the world's tin-production, something better will have to be found than what I saw in 1904.

DISCUSSIONS.

Chlorination of Gold-Ores; Laboratory-Tests.

Discussion of the paper of A. L. Sweetser, *Trans.*, xxxviii., 236.

Charles H. White, Harvard University, Cambridge, Mass. (communication to the Secretary*):—Those interested in leaching-processes gladly welcome contributions on either laboratory-tests, such as are reported by Mr. Sweetser, or mill-tests made on larger quantities of ore; and especially valuable are such reports, even with negative results, if the methods adopted are the outgrowth of an intimate knowledge of the conditions to be met. It seems very desirable, therefore, that as complete details as possible be given in reporting on such tests, particularly with respect to the character of the ore under examination. The questions of prime importance in the sampling of ore and in the making up of charges for laboratory-tests are those respecting its grade and uniformity. In Mr. Sweetser's paper the first of these questions is answered definitely, but the answer to the second, although apparent on inspection of the results, might be more strongly emphasized with advantage, since it is only on account of the extreme uniformity of the ore that the satisfactory results obtained could have been expected. It seems advisable to call attention to the fact that for average ores, and more particularly for uneven, or "spotted" ores, the quantities used by Mr. Sweetser are not usually regarded as sufficient to insure satisfactory results.

Let us examine the method somewhat in detail. The ore was crushed to pass a 5-mesh screen, and a sample of 45 lb. was taken out for assay and tests. The diameter of the largest grain of this ore, according to Richards,[1] was 2.67 mm.—assuming here, as in the following cases, that the wire of the screen was of the largest size used for the purpose, thereby giving any possible advantage in this respect in favor of the method. The table in Richards's book, p. 852, shows that 45 lb. is a sufficient quantity for a fair sample with uniform ores, but not sufficient

* Received Nov. 27, 1907.

[1] *Ore-Dressing*, vol. ii., p. 1196 (1903).

if the ore is uneven or of high grade. The 45-lb. lot was then riffled down without further crushing to a sample for assay of not more than 5 lb., which would be sufficient only in case of "very low grade or very uniform ore." The remaining 40 lb. was divided into four 10-lb. lots, which were crushed to pass 10-, 12-, 20-, and 30-mesh sieves respectively. According to Richards, the maximum grain in each case is respectively 1:35, 1.07, 0.69, and 0.44 mm., and his Table 369 shows that, with the most favorable ore, it is only the 30-mesh material that is sufficiently fine for so small a sample as 100 g.—the amount used for the tests—to be a representative portion. Notwithstanding this, Mr. Sweetser reports in Table III. the assays of four different lots of tailings which agree exactly with the assay of the original ore. The conclusion is therefore drawn that his ore was of rare uniformity, and this fact, no doubt, accounts in a large measure for his satisfactory results.

In dealing with such small quantities it seems unsafe to calculate the percentage-extraction from the assay of the tailings alone unless the ore is absolutely uniform. It is obvious that, with average ores, correct results by such a method could hardly be expected. It might easily happen that the tailings would assay higher than the original ore, and yet there might be a fair extraction. It seems to me much safer to assay the solution also, and thus have a check; the assay to be made, not by precipitating the gold as sulphide and filtering, as is suggested, but in the manner usually practiced of evaporating in a dish with litharge and assaying the residue, or evaporating in a glass beaker or dish of lead-foil, and running the residue through the furnace with its container. In dealing with tailings alone, it should be borne in mind also that there is a possible addition to the tailings of calcium sulphate from the charge, and also a possible shrinkage of tailings through the solution of other constituents than gold.

I think Mr. Sweetser is hardly justified in his statement that the fines were "obviously thoroughly leached," for in Table III. it is shown that the highest extraction was from the coarsest material, and that from the finest material nothing was extracted.

The peculiar result in Table II., that the finer the crushing the poorer the roast, may be explained on the assumption that

the finer ore bedded and was not stirred. Another interesting result set forth in Table IV. is that the longest treatment—5 hr.—gave the poorest results, and that the 4-hr. tests were worse than those of 1 hr. In these tests the fact that the ore was not thoroughly roasted might account for the precipitation of part of the gold after solution, if the quantity of chlorine present were not sufficient. Unfortunately, we are not able to estimate the chlorine pressure, since the capacity of the bottle is not given nor the quantity of water used in the charge, even if we could assume that the bleaching-powder was fresh and up to the usual standard in chlorine-content.

In the statement of the analysis of the ore, one of the constituents given is ferrous sulphide (FeS); also, there is an undetermined 3 per cent. in the report of the analysis, which is not mentioned.

The Vein-System of the Standard Mine, Bodie, Cal.

Discussion of the paper of R. Gilman Brown, *Trans.*, xxxviii., 343.

H. W. Turner, Portland, Ore. (communication to the Secretary*):—Mr. Brown's paper on the complicated groups of veins exposed in the workings of the Bodie mine forms a valuable contribution to the literature of the structural features of ore-deposits. It is much to be desired that managers should put into print the results of their observations of the mines under their charge. These observations, as well as data of production, kind of ores, etc., usually accumulate for years, and unless the mining geologists of the Geological Survey happen along before the mine is worked out, such data seldom get into print.

About the year 1899 I went through the mines at Bodie with R. C. Turner, then superintendent, and made some notes on the veins and gangue-minerals, and collected some specimens.

Mr. Brown refers to crystals of feldspar (albite) found on fragments of quartz on the surface. I collected a number of specimens of these crystals, mostly from Queen Bee hill, and

* Received Mar. 23, 1908.

had thin sections made of them for microscopic study. To my surprise, the examination of these thin sections showed that in all the sections the original mineral had been replaced by quartz. In other words, these crystals were pseudomorphs. As a study of the crystal forms with a goniometer was not made, I did not ascertain their original nature, but presumed them to have been feldspars. It will be noted that Mr. Brown likewise calls attention to the fact that some of these crystals were mere shells.

In a paper on Bendigo Veins,[1] Waldemar Lindgren describes the occurrence of albite in drusy cavities intergrown with quartz. He also calls attention to the occurrence of albite and a form of orthoclase in veins at other places.

Among the specimens of vein-matter collected at Bodie were some of banded chalcedonic quartz, showing thin seams of a clayey nature. It is quite possible that these clay-seams represent original feldspathic layers.

Under the head of dikes, Mr. Brown refers to the "Red Ledge" that cuts the Fortuna vein. I took a sample of this soft, sugary, dike-rock, and a portion of it was analyzed by Dr. W. F. Hillebrand, of the U. S. Geological Survey, who obtained the following results: SiO_2, 74.74; CaO, 0.14; Na_2O, 0.21; and K_2O, 10.57 per cent.

Evidently this dike is of the nature of an aplite or a rhyolite. My notes describe it as thoroughly granular, and if this is correct, the dike is an aplite. So far as I know, aplite-dikes in the Sierra Nevada have never been found intruded into Tertiary volcanic rocks. They are most common in granitic areas, but sometimes occur in the slates surrounding the granitic masses, as on the east slope of Mt. Conness, a few miles to the south of Bodie. It would therefore be interesting to have the nature of the "Red Ledge" dike determined with certainty.

The aplites of the granitic areas are regarded as allied to pegmatites, into which they grade, and to have filled contraction-cracks and fissures in the cooling granitic magma.

Since the basic elements in a given magma usually crystallize out first, the residuum is relatively richer in silica and the alkalies. This aqueo-igneous residual magma, filling cracks and fissures, forms aplite- and pegmatite-dikes.

[1] *Economic Geology*, vol. i., No. 2, p. 163 (Nov.-Dec., 1905).

In the old lower workings of the Bodie mines, there is often on the timbers and elsewhere an efflorescence of white silky fibers. This is magnesium sulphate, so common in old quicksilver-mines.

Secrecy in the Arts.

Discussion of the paper of James Douglas, *Trans.*, xxxviii., 455.

Edgar Hall, Silverspur, Queensland, Australia (communication to the Secretary*):—Dr. Douglas has written on this subject before, and his influence has had a powerful effect on the side of freedom among the younger generation of engineers. Moreover, it is not the first time that he has attributed the decline of the copper-smelting industry in Swansea to the secrecy maintained by the firms there engaged in smelting-operations.

In this latter statement I think his conclusions are open to criticism. Whether the firms mentioned have suffered by their secrecy is their own business—I hold no brief for them; but that the decline of the copper-industry at Swansea was inevitable seems to me to be proved conclusively by Dr. Douglas's own writings.

In his paper on The Copper Situation,[1] Dr. Douglas says:

"In comparing iron and copper mining, . . . it must be remembered that 2 tons of average iron ore from the most prolific source now available . . . will make 1 ton of metal; whereas from 33 to 75 tons (say an average of 50 tons) of copper ore must be mined to produce 1 ton of copper." And further: "Last year's production of iron was made from approximately 50,000,000 tons of ore. Our copper mines, to yield a production of 900,000,000 pounds, must have handled approximately 22,000,000 tons of ore, or nearly half the quantity raised from our iron mines."

Is not this the key to the whole situation? In transporting the 50,000,000 tons of iron-ore, 25,000,000 tons of the metal iron were carried so much nearer to the market, but the transportation of 22,000,000 tons of copper-ore would have carried only 450,000 tons of the metal copper nearer to its ultimate destination.

The small percentage of copper available in its ores makes

* Received Feb. 25, 1908.

[1] *Engineering Magazine*, vol. xxxiv., No. 1, p. 4 (October, 1907).

smelting at, or very near, the mines an absolute necessity, and modern pyritic smelting makes smelting at the mines still more desirable, owing to the ore furnishing its own fuel to so large an extent.

Swansea, in its palmiest days, never did smelt what is now considered low-grade ore, and never could hope to do so, even with all the cheap ocean-carriage of the past 25 years. Only rich ores and mattes could pay the cost of carriage, low as it has become, and, with the introduction of the Manhès Bessemer process, even matte can be more cheaply treated "on the spot."

As to electrolytic refining, probably the fact that the United States is such an enormous consumer of copper, together with another reason, to which I will allude later, has more to do with the limitation of the industry at Swansea than has the secrecy and lack of progress attributed to the smelting-firms there operating.

The other reason is in the ownership of the mines. At Swansea the industry was essentially a "custom" one. To-day the great copper-smelters own the mine, and it matters little who knows what is being done, for those who may copy their methods cannot take the mine, and it is decidedly of advantage to the smelter to know what others are doing, since every improvement adopted is pure gain and an increase in the value of the mine.

But to the custom smelter very often the possession of a "wrinkle" means the difference between profit and loss, and in the case of the Swansea firms, if they showed their methods freely, it meant that they would be promptly deprived of their supply of raw material.

No copper-mine owner will sell rich ore, or matte, or blister-copper to Swansea, if he can treat it himself, and the transportation-costs make it absolutely certain that as soon as he "knows how" he will so treat it. Could the Swansea firms be expected to teach him "how," when it meant taking away their living? I think they could not be expected to do so, and moreover, in their case, no exchange of knowledge could result in any important gain to themselves.

Dr. Douglas's references to the freedom with which the British iron-industry is open to visitors, and to the like openness

at the Rio Tinto works, and the Cape copper-works, in South Wales, of the respective mining companies, only confirm my argument. No copying of methods could deprive Great Britain of her iron-ore supply so long as her maritime supremacy lasts, and by no effort can we imagine the owners of the Rio Tinto or Cape copper-mines suffering from others learning their smelting-methods. Like the owners of copper-mines in America or Australia, they could only gain by knowing what other smelters are doing.

Electrolytic refining at the works by the mine-owners, individually or in groups, followed as a matter of course, the financial reasons connected with the fluctuations of the copper-market making the refining of the metal by the owners still more expedient.

For 20 years I have been a reader of Dr. Douglas's papers, with much pleasure and profit, and I am surprised that he should attribute the decline of the copper-industry of Swansea to "self-conceit and the inbred habits of trade-secrecy" so emphatically as he does. I think he is unnecessarily severe when he says that "she has preferred to gloat over her secrets behind closed doors rather than go out into the world in search of new business."

Incidentally, I fancy he is wrong in stating that New Zealand copper goes to America to be refined; if it does, it must be in small quantity, as hitherto New Zealand has produced but little copper. Probably he refers to Australian copper, and even this copper to some extent is now being refined electrolytically in Australia. New Zealand is 1,200 miles from Australia, and has "no connection with the party across" the Tasman sea.

Pure Coal as a Basis for the Comparison of Bituminous Coals.

Discussion of the paper of W. F. Wheeler, *Trans.*, xxxviii., 621.

A. Bement, Chicago, Ill. (communication to the Secretary*):—Formerly it was the general practice of engineers to designate coal that is free from moisture and ash as "combustible," notwithstanding that it contains oxygen and nitrogen. The origin of this practice was more particularly due to the requirements in connection with boiler-testing, which demanded a consideration of that portion of the fuel which took part in the combustion-process because results based on a unit of either moist or dry coal were not comparable, since both moisture and ash were recognized as variable quantities. Thus, a pound of "combustible" became a standard of value, and was treated as a constant before employing measurements of heating-power and calculating therefrom the percentage of efficiency. The recognition of this term "combustible" was an advance to be credited to the engineer rather than to the chemist, because the latter, not having to do with the interpretation of analyses in practice outside the laboratory, usually saw no reason for reporting results in terms other than those of the moist-coal composition.

Some years ago I needed a table of fuel-composition in terms of moist, dry, moisture-free and ash-free coal, and of the combustible components. This table required the use of the word combustible in its proper sense, for the purpose of designating carbon, hydrogen, and sulphur, instead of including these three elements with oxygen and nitrogen, to which aggregate the term combustible was formerly applied. To make such a tabulation involved either the use of the inconvenient and cumbersome term, "ash- and moisture-free coal," or of some other representative term, and for this purpose I adopted the expression pure coal, which is short, simple, convenient, and technically correct. It is also harmonious, because it agrees

* Received May 2, 1908.

with other designations of coal, such as moist coal and dry coal. I believe, however, that the term in question was suggested or used previous to my adoption of it, although I am not positive on this point.

The use of the term pure coal has received some criticism, and Prof. William Kent considers the expression moisture- and ash-free coal to be preferable. It is, however, favored by Prof. L. P. Breckenridge and many other engineers, and has come into general use. George H. Barrus argued that it is an incorrect expression, because he believes sulphur to be an impurity. If there were any coal free from sulphur it would be permissible to consider this element as an impurity, but as all coals contain it and in combination with other ingredients, it necessarily follows that it is a portion of coal proper, as well as one of the constituents of the ash associated with the coal. Thus pure coal is the ultimate conception of the composition as coal, with which moisture and ash are associated in what may be considered a physical rather than a chemical union. Any conception which separates the ingredients of the pure coal destroys it and divides it into its elements. There would be more cause for classing nitrogen as an impurity, because of its being considered as non heat-producing, than sulphur, which does evolve heat. Moreover, from this standpoint, hydrogen as compared to carbon could be regarded as an impurity because it burns to water instead of dry gas, and for this reason its products carry away proportionately more heat in the waste gases than do the products of carbon.

Criticism has been offered by chemists, due rather to the introduction of a new and unfamiliar feature than to any real opposition to the idea of pure coal as a factor. Yet it is gratifying to observe that the term is being accepted, and that such a valuable discussion as that of Mr. Wheeler has been published.

At the time I suggested the use of the heating-power of the pure coal as a factor, it was thought that this factor would be applicable to a definite district or field, although it was known that it would not apply to a large territory, such as the two counties of Williamson and Franklin in Illinois. It was also recognized that different grades of fuel, such as lump-coal, mine-run and screenings, would no doubt have a characteristic pure-coal value of their own. Thus, if the locality of occurrence

be known, and ash and moisture be determined, the use of this factor would enable a calculation of heating-power of the moist or dry coal to be made without a calorimetric determination, a scheme which would apply with definite exactness in the valuation of fuel-coal, in which the variation of heating-power between the top and bottom of the seam, or between different benches, would have no effect, because the seam is mined and shipped as a mixture.

Recently, however, the matter of the use of pure coal as a constant has received much attention by members of the chemical departments of the University of Illinois and the Illinois State Geological Survey, with the result that two new features have been offered. The first of these is that certain corrections should be made on account of the presence of sulphur. From the stand-point of fuel-coal, sulphur has always been an element of confusion to chemists, and some are disposed to regard it as not being entitled to consideration as a heat-producing element. Inasmuch, however, as it does contribute in a measure to the generation of heat, this position is untenable and no further discussion concerning it is necessary.

The second feature, which Mr. Wheeler calls water of composition of the ash, meaning the "loss on ignition," used, for example, in connection with cement- and clay-analyses, is now attracting attention in Chicago, and the idea is that a certain portion of the ash is volatile, and therefore dispelled during the process of burning in ash-determination, so that the remaining matter, instead of being the total, is what may be termed a residual ash. Thus it is this total ash which excludes the heat-producing matter in the fuel, and as the heating-power determinations are at least made on dry coal, it necessarily follows that the pure-coal value as obtained by calculation must be increasing by an amount equal to the ash found, but inasmuch as thus far the only source of information as to the actual quantity of ash is given by the residue rather than the actual total ash, the heating-power based on the residue will, of course, be inaccurate. So in those cases in which a portion of the ash is of a volatile nature, it follows that the larger the percentage of ash the greater will be not only the amount of volatile matter, but also its percentage of the whole quantity of fuel. If the ash in the fuel consists largely of dirt mixed with the coal,

such as fire-clay, the volatile portion may be quite high. Following this reasoning, which seemed to apply frequently in practice, it appeared that the dirtier the coal the lower will be the calculated pure heating-value. Very clean coal will show high heating-value of the pure composition, while dirty coal will indicate a low one.

It has been proposed to establish dry-heat values on the basis of the percentage of ash found, so that a determination of heat-value would be unnecessary if the source of the fuel were known. And therefore a determination of ash on the basis of dry coal is all that would be required.

The foregoing remarks are on the assumption that there will always be a loss by volatilization, so that the residual ash will be smaller than the real ash. This assumption, however, does not always apply; for example, with ash high in iron-content, the residue, owing to oxidation, will actually weigh more than the real ash. This complication refers the matter back to my original proposition—namely, the establishment of pure-heat values for localities or seams, or localities in seams, and for different grades of coal. Mine-run coal would contain more dirt than lump-coal; likewise, screenings would be higher in ash than mine-run, and each class would have a different pure-coal value, obtained on the basis of residual ash, it being quite immaterial whether or not the residual ash is the real or the correct one, since it is that with which the customer deals in practice. The volatile portion goes up the chimney; it does not trouble the fireman in its removal from the furnace, or the user in expense for cartage, neither does it interfere sufficiently with the combustion-process to be considered, because the use of the values as constants will give correct results as affecting the dry- or moist-coal composition with which the user deals. Tables I. and II., and subsequent examples, are presented to illustrate this statement, it being assumed, for purpose of illustration, that volatile loss is 7 per cent. of the real ash in Table I., and the weight of the ash in Table II. is increased a like amount.

The data in Tables I. and II. show that the pure B.t.u. factor, as determined from the residual ash, is lower than it should be with ash of such character that a portion volatilizes and is lost; and higher in those cases in which the residual

TABLE I.—*Decreased Residual Ash.*

Example.	Actual Pure. B.t.u.	Actual Dry as Determined. B.t.u.	Ash. Total. Per Cent.	Ash. Residual. Per Cent.	Apparent Pure, Calculated from Residual Ash. B.t.u.
No. 1. 10 per cent. ash basis, . .	14,500	13,050	10.00	9.30	14,388
No. 2. 15 per cent. ash basis, . .	14,500	12,325	15.00	13.95	14,323

TABLE II.—*Increased Residual Ash.*

Example.	Actual Pure. B.t.u.	Actual Dry as Determined. B.t.u.	Ash. Total. Per Cent.	Ash. Residual. Per Cent.	Apparent Pure, Calculated from Residual Ash. B.t.u.
No. 1. 10 per cent. ash basis, . .	14,500	13,050	10.00	10.70	14,613
No. 2. 15 per cent. ash basis, . .	14,500	12,325	15.00	16.05	14,681

ash becomes heavier than the real ash. The following examples serve to show that the pure coal factor as obtained upon the basis of residual ash gives correct results in the calculation of the heating-power of dry or moist coal.

Example No. 1, with a determined actual dry heating-value of 13,050 B.t.u., assuming that the actual or total ash is known, shows that the pure value is 14,500 B.t.u.

$$(1) \qquad \frac{13{,}050 \times 100}{100 - 10} = 14{,}500 \text{ pure B.t.u.}$$

Example No. 2, wherein the residual ash is used, gives a pure value of 14,388 instead of 14,500 B.t.u.

$$(2) \qquad \frac{13{,}050 \times 100}{100 - 9.3} = 14{,}388 \text{ pure B.t.u.}$$

But the following example, (3), shows that the use of this factor, 14,388, nevertheless gives the correct B.t.u. per pound of dry coal:

$$(3) \qquad \frac{14{,}388 \times (100 - 9.3)}{100} = 13{,}050 \text{ dry B.t.u.}$$

Moreover, from the standpoint of ash which instead of decreasing in weight becomes heavier, examples (4), (5), and (6) illustrate the process in a similar manner, using values, of course, which appear in Table II.

$$(4) \qquad \frac{12{,}325 \times 100}{100 - 15} = 14{,}500 \text{ pure B.t.u.}$$

$$(5) \qquad \frac{12{,}325 \times 100}{100 - 16.05} = 14{,}681 \text{ pure B.t.u.}$$

$$(6) \qquad \frac{14{,}681 \times (100 - 16.05)}{100} = 12{,}325 \text{ dry B.t.u.}$$

One of the difficulties in discussions of this kind is the disposition of different parties to look at the subject from their individual view-points. The chemist often regards the matter from the stand-point of the laboratory rather than that of coal-production and utilization. It is well to bear this point in mind, and to insist upon a consideration of the proposition in its application to the particular phase of the problem to which it applies.

In the foregoing, I have been concerned entirely with the problem of the simple and intelligent treatment of fuel-coal of a known locality of production, but the idea becomes much more complicated when it is necessary to identify the fuel by means of analysis. It is in this latter connection that suggestions like those of Mr. Wheeler have value. I wish to call attention, however, to the great complexity of the matter, and to urge that conclusions be not hastily formed, since it is difficult to devise any simple method which will give information practically equivalent to that of an ultimate analysis. I do not propose to use the pure-coal factor as a basis for the comparison of coal in any such exact sense as implied in Mr. Wheeler's paper.

The Corrosion of Water-Jackets of Copper Blast-Furnaces.

Discussion of the paper of George B. Lee, *Trans.*, xxxviii., 877.

C. D. Demond, Anaconda, Mont. (communication to the Secretary*):—In order to throw some light on this interesting subject, a series of experiments were made with strips of mild steel, containing about 0.14 per cent. of C and 0.22 per cent. of Mn. These strips were thoroughly cleaned and brightened before use, salts were added to distilled water until it corresponded as nearly as possible to the analysis given by Mr. Lee, and in this water at different temperatures were placed four strips of steel; while four other strips, at corresponding temperatures, were placed in the water used at the Washoe Reduction Works. The latter plant has had no trouble from corrosion of furnace-jackets, pipe-lines or boilers.

The analysis of the water used at the Washoe works is:

	Grains per U. S. Gallon.
$CaCO_3$	2.92
$CaSO_4$	0.76
$MgCl_2$	0.64
$MgCO_3$	0.58
Al_2O_3	0.12
Fe_2O_3	0.12
SiO_2	0.58
NaCl	0.35
Organic matter, etc.,	1.16
	7.23

This water is neutral to litmus; while the artificial Copper Queen water was very slightly alkaline, which Mr. Lee informs us is also true of the water at Douglas.

Both waters were well aërated by pouring from beaker to beaker, and by blowing in air from the experimenter's lungs. The results of these tests, given in Table I., show that there is no significant difference between the effects of the two waters.

* Received July 25, 1908.

TABLE I.—*Oxidation of Iron Per Sq. In. of Surface* When Immersed in Certain Waters.*

Average Temperatures.	First 50 Hours.	Second 50 Hours.	Total.
	Artificial Copper Queen Water.		
	Mg.	Mg.	Mg.
70° F.	1.943	1.360	3.303
106	5.322	3.963	9.285
150	5.206	4.701	9.907
185	2.865	1.642	4.507
	Washoe Water		
70° F.	2.797	1.516	4.313
106	5.178	3.746	8.924
152	6.618	6.422	13.040
188	1.784	0.699	2.483

* These results were obtained by carefully rubbing all the rust from the iron, dissolving it, and determining its quantity by titration.

At two temperatures the strips of steel in the Copper Queen water showed less rust than those in the Washoe water, while at the other two temperatures they showed more. Another experiment indicated that distilled water would have given about the same results. Hence, the corrosion of the Copper Queen jackets must be due to some other cause than the quality of water. I am informed, however, that the water is raised from artesian wells by means of compressed air, and, after being used in the furnace-jackets, is passed over a cooling-tower, and later goes to the jackets again. Hence it probably contains an excessive amount of dissolved oxygen. It will be noticed that rusting increased with the temperature, up to 150° F., as is true of chemical actions in general; but at 185° there was decidedly less rust than at 150°, presumably due to the water at the higher temperature being unable to hold enough oxygen in solution to keep up the rate of oxidation.

The small differences of temperature (9° to 14° F.) do not seem sufficient to account for the inner plates rusting more than six times as fast as the outer; but is it not probable that the temperature of the very surface of the inner plates is considerably higher than is shown by inserting a thermometer in the water, and that an unusual amount of dissolved oxygen at this temperature is the cause of the trouble? The jackets would probably suffer much less if the water were discharged as near

the boiling-point as practicable, and returned directly to the supply-tanks; the return-pipe should dip well beneath the surface of the water in this tank, since a plunging stream would entrain a fresh quantity of air. This would lessen the amount of dissolved oxygen, and rust cannot form without oxygen. Referring to the first two paragraphs of Mr. Hixon's discussion,[1] it is the gases in solution that are active, not those given off; and the probable reason for his jackets corroding most rapidly opposite the cold-water inlet is that the water at that point still holds the gases in solution. The absence of rust in the Copper Queen boilers, and in those at the Victoria mines, is perhaps due to the elimination of absorbed gases immediately on entering the boilers and before the water can touch the metal. It is certain that the temperature of the water increases very rapidly upon entering the boilers.

We have experimented with several other remedies. The well-known use of zinc is completely effective when the zinc is properly distributed; but this metal corrodes so rapidly that it has to be frequently renewed. Copper, on the other hand, increases the rusting of the steel, these results being due to the electro-chemical relations of the metals.

Knowing that solutions of alkalies and alkaline salts sufficiently strong will prevent rusting, we tried the effect of lime, at approximately the temperature of the water in the Copper Queen jackets. An increase in the quantities of lime added up to 0.1 per cent. gave increased benefit, amounting to practically complete prevention with the Washoe water, and reducing the rust in the artificial Copper Queen water about 60 per cent. The difference was evidently due to the reaction of the lime with the sulphates and carbonates in the latter water. Larger amounts of lime would probably complete the cure, but the precipitate might have to be settled out.

Dr. A. S. Cushman[2] found, in certain experiments, that potassium bichromate completely prevented rust when dissolved in water at the rate of 1 lb. or more in 1,500 gal. Our tests indicate that, even in using the water over and over in the jackets, the bichromate would be slowly exhausted, requiring fresh addi-

[1] *Trans.*, xxxviii., 882 (1908).

[2] *Bulletin* No. 30, *Office of Public Roads, U S. Department of Agriculture* (1907); and *Proceedings of the American Society for Testing Materials*, vol. vii., p. 211 (1907).

tions at intervals. We have not yet determined what the cost of treatment would be; but the protective effect is very striking.

The elimination of dissolved oxygen from the water is suggested as the practical remedy in this particular case; but we may take occasion, here, for some further remarks on the general subject of rusting.

The fact that some wrought-iron pipes at Douglas are practically unaffected, while others in the same line, as well as the furnace-jackets, are badly corroded, strongly suggests that there is some trouble with the metal itself, and that it is not a question of wrought-iron versus steel. Indeed, the most recent and reliable industrial and scientific investigations show that the long-standing controversy as to the relative rust-resisting qualities of wrought-iron and steel, is largely a beating of the air. The real question seems to be one of care in the processes of production. Cushman[3] very ingeniously demonstrates that an apparently homogeneous piece of iron or steel carries a multiplicity of positive and negative poles of an electrolytic system; and the electrolysis between them increases the speed of rusting at the positive pole, while preventing rust at the negative. He suggests that this polarity is due to uneven distribution of certain chemical constituents of the metal. In answer to the fact that frequent investigations have failed to show this uneven distribution, he says[4] that "such extremely small differences in the chemical composition as might easily escape detection in ordinary chemical analysis are still sufficiently large to account for slight differences of electrical potential." He might have added that there may be considerable difference in composition which chemical methods cannot detect, because it is impossible to sample the segregations separately. Frank N. Speller,[5] on the other hand, thinks that variations of density may sometimes account for the polarity. However, there is no doubt as to the existence of the polarity; and its effect was well shown in one of our test-strips, 4 in. long, after an exposure of 50 hr. in water. A large part of the surface shows perfectly bright, but other parts are

[3] *Loc. cit.*

[4] *Farmer's Bulletin No.* 239, *U. S. Department of Agriculture*, p. 20 (1905).

[5] Applied Science, *Proceedings of the Toronto Engineering Society*, Jan., 1908, p. 125.

badly rusted. Mr. Lee says "the plates are pitted and eaten away in some places more than in others;" also, that in one case "there was a place about 6 by 8 in. right in the middle of the plate which seemed to be quite smooth and not pitted at all, while all around it was very badly corroded." The greater the difference of potential the greater will be the action at particular spots, which will increase the pitting and quickly produce holes in one piece of metal, while another piece, of the same size and thickness, long remains serviceable, even if yielding the same amount of rust in a given time, because the action is uniformly distributed. This action of iron is similar to that of zinc. The latter, when pure, dissolves very slowly in acid, but when impure it dissolves readily, because of the electrolysis between the spots of pure zinc and the spots of impurities.

The fuller details of this explanation accord with the present-day theory of physical chemistry. Cushman's results confirm earlier work by Whitney, and are independently verified by Walker.

There may be many hundred independent local circuits on a few square feet of surface. Cushman[6] says that "in almost all modern steel woven-wire fences some wires will be found to far outlast others, . . . It is just this point of unevenness of lasting quality in wires from successive heats in the same mill, which have practically the same chemical composition, that is hard to explain by any theory but that of galvanic or electrolytic action."

These ideas suggest the use of better steel for furnace-jackets; though the fact that the rapid corrosion has occurred with jackets purchased from at least ten different manufacturers[7] makes it appear that better metal is hard to obtain. Moreover, in view of the satisfactory service of the outer plates, it may seem, at first thought, that the contention for better steel is not well made. But the key to this difficulty probably lies in the different histories of the inner and outer plates in the steel-mill.

For years the manufacturers have fully appreciated the practical benefit of an increased amount of work put upon steel,

[6] *Farmer's Bulletin No.* 239, *U. S. Department of Agriculture*, p. 21 (1905).

[7] Private letter from Mr. Lee.

at the proper temperature, in improving the physical properties. The National Tube Co. has found that the resistance to rusting is greatly increased by similar treatment, and this company has developed a special method of applying it.[8] Now the rolling of the outer plates of the water-jackets to a thickness of $\frac{3}{8}$ in. necessarily requires more work than rolling the inner plates to 0.5 in., and we may therefore expect the former to be more resistant to rusting. The angle-iron stiffeners, which Mr. Lee says do not rust seriously, probably received a good deal of work. The fact that stay-bolts rusted badly at the ends next to the inner plates, while being little affected at the other ends, is explained by the electrolytic theory as follows: It is well known that rust, once formed, increases the rate of corrosion. Mr. Speller[9] reports finding a potential as great as 50 millivolts between one clean iron rod and another rod exactly similar which had a very slight coat of rust, both being immersed in water. When both rods were clean, the potential was much less. The rust on the inner plates of the water-jackets will set up a current between itself and a stay-bolt, which will corrode the latter. This current, in seeking the path of least resistance, will pass from the stay-bolt through the water and back to the rust of the inner plate without going as far as the outer plate. This effect will be decidedly less in the case of angle-iron stiffeners, because the latter do not have the intimate contact with the inner plate that the stay-bolts have, and therefore the electric current meets more resistance.

Speller found the voltage between steel and mill-scale to be greater than between the metal and ordinary rust. It may be that, for some peculiar reason, this scale is always worse on the 0.5-in. than on the $\frac{3}{8}$-in. plate, though this is hardly probable. However, it is advisable to see that all scale and rust are thoroughly removed before the jackets are put together. I believe that the main point is the working of the metal in the mill. The steel-manufacturers are making important investigations with valuable practical results, but if they do not yet supply 0.5-in. steel of suitable quality, it may be well to try jackets made wholly of $\frac{3}{8}$-in. stock.

[8] Private letter from Frank N. Speller of the National Tube Co.

[9] Applied Science, *Proceedings of the Toronto Engineering Society*, Jan., 1908, p. 125.

Arthur S. Dwight, New York, N. Y. (communication to the Secretary*):—Several times in the course of my smelter-work have I experienced trouble from corrosion of blast-furnace water-jackets, particularly of the wrought-iron or steel jackets commonly used in copper-smelting, and in a manner very similar to that described by Mr. Lee, but, in my case, always traceable in the end to acid water.

The troubles at Douglas seem to be somewhat more aggravated than one usually encounters, and the causes more obscure. I understand that, like most of the smelters in the Southwest, the Copper Queen works has two distinct systems of water-pipes, one for the regular high-pressure service, distributing fresh water to all parts of the plant, and for fire protection; while the other, at low pressure, circulates the jacket-water between the blast-furnaces and a system of cooling-towers. The water in the jacket circulatory system has numerous opportunities to pick up soluble sulphates, principally in the vicinity of the blast-furnaces (where it is almost impossible to prevent copper flue-dust from sifting into the launders and open places in the circuit), or by flue-dust being blown into the waters of the ponds and cooling-tower. Although the tests for acidity in the jacket-water which Mr. Lee has made from time to time have failed to show the presence of acid, and although we may perhaps properly suspect some other cause to be the principal one, I am strongly inclined to think, nevertheless, that the troubles are aggravated by the presence of acid in the water, particularly as it seems to be admitted that the corrosion is somewhat more evident in the circulatory system than it is in the service system.

The fact, however, that some pitting and corrosion also occur in the fresh-water line points to the presence of some peculiar property in the water itself. The irregularity with which some sections of pipe and some jacket-sheets are corroded, while others in the vicinity, or even adjacent parts of the same piece of metal, are entirely free from attack, would point to faulty material, or perhaps careless heat-treatment in the manufacture of the steel. But, granting all this, we are still far from a satisfactory answer to the riddle. The Copper

* Received Sept. 7, 1908.

Queen smelter is built of the best materials currently obtainable, purchased from various makers, and, as it stands, represents better than average modern smelter-construction. The analysis of the water gives no clue; in fact, it must be said to be unusually harmless-looking. The theory of electrolysis does not seem to meet the conditions. By the process of eliminating those theories which fail to satisfy, we are finally confronted with the question: can the fact that this water is raised from the deep wells by compressed air have any bearing on the problem? Is it not possible that, by the intimate commingling of the water and air at high pressure, such thorough aëration might occur as would make the water an extraordinarily active oxidizing agent? This could easily be determined by experiment. Personally, I should consider this a surprising fact if it proved to be the correct explanation, but the logic of the situation seems to point to some such cause out of the ordinary.

In this connection, it may be pertinent to give some of my own experiences with troubles of this kind, and the remedy that was developed.

The most serious case of corrosion of pipes and water-jackets I have ever had occurred while I was in charge of the operations of the Greene Consolidated Copper Co., at Cananea, Sonora, Mexico, especially during the latter part of 1905. Cananea is not more than 60 miles distant from Douglas, and there is some similarity in the conditions of water-supply to the furnaces and in the character of the ground-water, etc.

Our troubles were made the subject of a long series of studies by R. L. Lloyd, who was then superintendent of the smelting-department, and to him is due the credit for finding a practical and satisfactory remedy. It seems to me proper, therefore, that he should be allowed to tell it in his own way, and I insert the following extract describing the episode in detail from a letter written by him at my request, and which I have his permission to include in this discussion:

Complying with your request to give you the details of the manner in which we worked out the troubles at Cananea in connection with the serious corrosion of the water-jackets and pipes in the circulatory system of the blast-furnaces, I take pleasure in giving you the facts as I can recall them, though I am at a disadvantage in not having my notes on the subject accessible. I agree with you in thinking that the troubles which Mr. Lee describes as being so serious at the Douglas plant

are almost identical with those which we experienced, and I think it very probable that the same remedy may correct his difficulties also

At Cananea, the main part of the trouble occurred in the long pipe-lines which extended from the water-cooling tower to the jackets. There was also trouble in the jackets, particularly at the joints around the tuyeres. It was noticed that wherever iron and brass were in contact, as, for instance, at the brass valves, the corrosive action was greatly intensified, presumably on account of local electrolysis. The trouble at one time assumed such proportions that it became very difficult indeed to keep the pipe-lines and jackets in sufficiently good repair for steady operation, and many expedients were tried in the attempts to discover a remedy. Slaked lime was used to neutralize the acid in the water, but the results were only partly successful, and were attended with the serious disadvantages arising from the accumulation of the lime in the tanks, and the formation of lime scales on the smaller pipe-lines. We then tried crude sodium carbonate, such as we had been using to prevent scale in the boilers, which gave better results than the lime, but was still far from satisfactory. Large slabs of zinc were also connected up in various places, especially in the steel overflow-tank, in the attempt to counteract electrolysis.

After much study and experiment, we finally determined the primary cause of the acid in the water to be the absorption of fumes of SO_2 by the sprayed water in the cooling-tower, which was situated on the leeward side of the smelter with respect to the prevailing winds and on top of a hill, nearly on a level with the top of the furnace-building. The SO_2 gas in the smelter-fumes blowing through the cooling-tower was dissolved by the falling water and slowly became oxidized, and eventually formed a dilute solution of sulphuric acid. The amount of corrosive sulphates was further augmented by the fact that more or less flue-dust got into the water-system by sifting into the launders around the furnaces and under the feed-floor, as is likely to happen at any smelter-plant, as usually constructed. The position of the cooling-tower was unfortunate, but on account of limitations of space it could not have been avoided, even if the trouble due to the smelter-fumes had been anticipated.

While working hardest to correct the difficulty, which threatened to be most serious, I read an article published in pamphlet form by an author whose name I have among my notes, but unfortunately not now accessible, which mentioned the effect of arsenic salts in deterring the solubility of iron and steel in acid solutions. I was at once very much interested to know if this could possibly have any bearing on this problem, and I proceeded to "doctor" the water-system with commercial arsenic oxide. The good effect of this addition was felt very quickly, and the corrosion was practically ended. From that time on, 1 kg. of arsenic oxide was added to the water-system each week, and a portion of the water in the circulatory system was allowed to run to waste, being replaced by fresh water, when analyses showed that the amount of sulphates was getting high.

In this manner we were able to avoid the corrosion of the jackets. It was noticed that even when the water became appreciably acid there was little or no trouble from corrosion of the pipe-system and jackets, though we always endeavored to keep the acid neutralized with commercial sodium carbonate.

It is to be regretted that Mr. Lloyd's citation of the article from which he obtained the suggestion for trying arsenic oxide cannot be made more concise at this time, but if it proves to be a matter of special interest it can doubtless be supplied later;

meantime, perhaps the paper may be known to some members of the Institute, who can complete the facts.

In this connection, it would be interesting to ascertain whether there is as much trouble experienced from corrosion of pipes and water-jackets in smelting-works treating large quantities of arsenical copper-ores, as there is in plants like Douglas and Cananea, which treat ores exceptionally free from arsenic. It is not impossible that the arsenic-fumes might automatically furnish the needed antidote for the acid in the water. Personally, I have no comparison which I can present from my own experience.

The following incident, which recently came to my attention, presents what seems to be an interesting confirmation of the deterrent action of arsenic in the corrosion of iron. A car-load of commercial sulphuric acid was purchased by a Western steel-works for pickling wire. The acid refused to work properly, though chemical tests showed it to be of proper strength. The steel company complained to the acid-makers, who sent an expert to investigate. He looked over the situation, promptly sent the lot back to the factory, and substituted a new lot of acid for it. Though inclined to be reticent about the cause of the trouble, the expert finally admitted to the chemist of the steel-works that this particular car-load of acid had been made from pyrites containing considerable arsenic, and that there was an appreciable amount of arsenic in the acid, stating, furthermore, that had his company known the purpose for which the acid was to be used, they never would have sent the kind they did.

Corrosion of jig-screens and other iron-work in wet-concentration mills might also be averted by applying the arsenic remedy, though I have never heard of its being tried.

J. A. Thomson, Victoria, B. C., Can.* (communication to the Secretary†):—In reference to this pitting and corrosion in the water-jackets of blast-furnaces, to my mind there is no mystery or fancied "electrolytic" action in the question. It is simply an effect of the air carried by water, fed to the jackets to keep

* Private communication to Francis A. Thomson, Pullman, Wash.

† Received Apr. 27, 1908.

them cool, and the action is as follows: As soon as the cold water comes in contact with the warm part of the jacket, it is heated and compelled to give up its air, which, being in contact with the plate, settles thereon. The circulation being sluggish, it is only when the bubbles have grown sufficiently large that they rise, and this rise is hindered to some extent by the bosh. During the period of rest, the air, containing both oxygen and carbonic acid, will attack the iron, and when small irregularities have been thus formed, subsequent bubbles find still better lodgment and speedily effect the formation of pit-holes.

If the water is fed to the jacket near the bottom, and if it is saturated with air, it can be shown that every square inch of heated surface of the jacket generates about 4.5 cu. in. of air per hour, equal to about one bubble $\frac{1}{8}$ in. in diameter per second.

The great difference of temperature between the water- and the fire-side of the jacket-plate, with the consequent straining of the grain of the plate, quickly loosens all rust as it is formed, so that metallic iron is always exposed to the air given up by the water.

The smooth parts mentioned as having escaped this pitting may be accounted for in a number of ways. At the start, such a spot may have had some adhering slag or other substance which reduced the heat at that point, so that the first irregularities produced would be formed away from that spot, and the bubbles would be more continously produced where they found good lodgment. Or, again, the spot may be a patch (of the kind found in all plates) which has been clean-rolled in the making. This sort of thing is a frequent occurrence in the pitting from similar causes on the furnaces of marine boilers.

The same reasoning, but reversed, will apply to the outside sheets. There is no extra heat on the outer side of the sheet, hence bubbles do not form there, but only on the hot sheet next the furnace.

As to the impellers of the rotary pumps, this case is similar to the action on a ship's propeller, in which the air is not driven out by the heat, but is abstracted by the partial vacuum formed, and in which, in spite of the high velocity of the water, there seems to be sufficient time for the mischief to be done. Even bronze blades are sometimes pitted in the same way.

My recommendation would be in the line of putting the water under pressure before admitting it to the jackets; that is, let the inlet to the jacket have a non-return valve like a large feed-check, with a stem carrying a weight, so that the pump feeding the jacket would have to force the water against, say, 20 or 30 lb. per sq. in. From under this check a small (say, $\frac{3}{8}$-in.) pipe should lead back to the well. The water being under pressure before entering the jacket, a large part of the entrained air will escape through this pipe. I do not mean the jacket to be under pressure, but only the pipe to the inlet. I have no doubt this arrangement would give much relief. I have used it many times as a remedy for pitting in marine boilers.

Of course, I presume there are no unknown acids in the water, and that it is just ordinary potable water, as seems to be proved by the fact that the boilers are said not to have been attacked in any way.

With regard to the statement of George M. Douglas, his experience in the White Star boats must have been decidedly limited, or he would not talk of a little salt water causing corrosion. It is a common practice to fill new boilers with sea-water at first, for the purpose of preserving them; and when I went to sea we had no evaporators to make fresh water, and if we ran short, we made up with sea-water. I have run a ship from Hankow, China, to London, 43 days' hard driving, without opening a cock that was not open at the start, and not a pint of fresh water to make up waste, only the sea to draw from, and at the end of the voyage the boilers were in excellent condition.

If a satisfactory way cannot be devised to introduce the water under pressure, I think that the use of nickel steel would solve the whole problem.

Piping and Segregation in Steel Ingots.

Discussion of the paper of Henry M. Howe, *Trans.*, xxxviii., 3.

P. H. Dudley, New York, N. Y. (communication to the Secretary*):—The characteristics of Professor Howe's metallurgical papers are, that he is able, from the mass of confusing evidence on the subject, to make an analysis and present hypothetical and essential principles in concrete form for consideration and discussion. I can appreciate, from the practical side of the subject, the labor of collecting and investigating the data which was essential to prepare the paper. Much of it necessarily came from material of preceding years of manufacture in which time was allowed for chemical reactions of the recarburizer, and for the elimination of the oxidation-products before teeming the ingots. This practice gave a favorable time-element to free the metal from slag and other impurities—a requisite condition for the production of sound steel in the ingots. But, in allowing them to become cold before they were charged into the furnace, a subsequent time-element entered as a detrimental factor in the development of the pipe or shrinkage-cavities, which gave an ingot-structure deficient in soundness, at least in the upper portion. (The term slag is used in a general rather than in a strictly technical sense.) Traces of manganese sulphide are often associated with manganese silicates in the same thread or seam, and at other places they are separate and distinct. Interpretations from material thus manufactured must be made with due allowances for the methods of fabrication and the requirements of service.

The only way we now have to estimate what the degree of soundness of the ingot-structure was, in the early foreign and American manufacture of steel rails, is by examination of the worn and failed material after years of service. We must remember also that the poorest material was long since consigned to the scrap-heap without any full record of its history. The rails from a given ingot cannot now be identified; and in many

* Received Feb. 18, 1908.

cases even the particular "heat" cannot be traced. It is thus evident that investigators labor under great difficulty in differentiating between reliable and unreliable data incident to variable methods of manufacture, chemical composition of the alloys of steel, mediocre practice and that of the highest state of the art.

Modern practice and large outputs in Bessemer steel have reversed the effects of the two time-elements: first, by cutting short the time for the chemical reactions of the recarburizer and the elimination of the oxidation-products before teeming; and, secondly, the full shrinkage of hot to cold metal has been checked by charging the ingots into furnaces after teeming, and blooming at least with the equalized original heat of the freezing metal.

This practice has reduced the great number of true pipes in the early steel rails from cold ingots—though some are still produced—while the partial elimination of the first condition has deteriorated the quality of the steel in the ingots, the upper interior portion being heterogeneous and unsound metal.

The segregation in steel which is not deoxidized sufficiently becomes concentrated and intensified in effects by blow-holes, and is more harmful in large ingots, which are kept in vertical position from the teeming to the blooming, than was the case with those of smaller dimensions, thrown down on their sides in the pits after stripping, laid on cars and charged into horizontal furnaces, for equalizing the heat for blooming. There are many defects of ingot-structure incident to the concentrated segregation surrounding the blow-holes of the central core of the ingot. Some of these blow-hole cavities contain a small button, a minute aggregation of deoxidation-products of the recarburizer, and when elongated and compressed by the blooming and rolling into the section for rails, form some of the minute seams with slag-inclusions.

When the columnar structure of the ingots contains as inclusions the deoxidation-products of the recarburizer either as manganese silicates—an olive-green slag—or manganese sulphide, the metal is usually "red-short," and the quantity of second-quality rails produced is large.

"Split heads" are due to included slag in the bearing-surface, and many of them are independent of any pipe in the

web, though there are combinations of blow-holes in the web with some split heads.

There are other defects of ingot-structure due to entrained slag, oxides, and gas, particularly at the junctions of the columnar structure in the corners of the molds in the upper part of the ingot. The darkening under traffic and then spawling of metal from the upper corners of the head are indications of deficient and unsound ingot-structure, which are quite distinct from general piping or segregation. The entanglement of impurities in the jutting pine-tree crystals of the freezing metal may form a region of unsoundness between an outer envelope of the columnar structure and the central core, which, in the bearing-surface of the head, becomes oftentimes separated at the joints of the upper rail of the ingot when under heavy traffic.

These defects of ingot-structure are not included in Professor Howe's paper, but should be in any discussion, in my opinion, where unsound ingot-structure from piping or segregation is under consideration.

The failures of rails are due to many causes, of which a number, such as flow of metal in the bearing-surface of the head, "mashing" of the ends, and flaking of metal from the upper corners of the head, are due to insufficient cubic elasticity or elasticity of volume of the metal to receive, sustain and distribute the passing wheel-effects to the rest of the section. The deficiency may be due to a low grade of metal or entrained slag, oxides or other impurities in a metal intended, from the chemical composition, to be of high grade.

Professor Howe's Introduction divides his paper into three parts: the first treats of the causes and the restraining of piping in steel ingots, the second considers the causes and the restraining of segregation, and the third proposes certain precautions in engineering specifications concerning these two defects.

In Part I. he says:

"I infer that the pipe is chiefly due to what I call the virtual expansion of the outer walls of the ingot in the early part of the freezing. I find that the upper and smooth-faced part of the pipe probably forms while the interior is still molten, but that the lower, steep, and crystal-faced part probably forms in metal which is already firm."

I have not been able to observe in a great many ingots which

I have cut to trace the pipe, that any of the lower part had opened in metal which had previously solidified. I find frequently in ingots from special composition that they pipe from the top nearly to the bottom, while in others, with care, the pipe is eliminated with from 5 to 6 per cent. discard. But, as stated previously, I have never found an instance in which the metal had separated after being firm, as inferred by Professor Howe, which, if it did not occur in a cold ingot, would probably not in one bloomed before it was allowed to cool. He designates five causes which may co-operate to limit the depth of the pipe, and finds that blow-holes, sagging, and the progress of freezing from below upwards are usually effective. He finds that the pipe may be lessened by casting: "1, in wide ingots; 2, in sand molds; 3, at the top instead of at the bottom; 4, slowly; 5, with the large end up; 6, by the use of a sink-head or other means of retarding the cooling of the top; 7, by permitting blow-holes to form; and 8, by liquid compression."

Professor Howe, writing in general, without entering into specific details, has not discussed the effect of chemical composition in the setting of the ingots, except in reference to Brinell's well-known experiments, though under 7, "by permitting blow-holes to form," there is some reference to chemical composition.

Brinell's experiments evidently were made upon smaller ingots than those generally used for rails the past two decades, for they need decided modification in this country, particularly with direct metal.

I have found that by modification of the chemical composition to suit the conditions at a particular mill, the setting of the steel could be improved, and in small ingots for three lengths of 30-ft. 100-lb. rails, sound ingots were produced, under the usual discard of from 6 to 8 per cent. from the top.

This is the day of specifications, the consensus of opinion being that those of the same chemical composition apply with equal force to any mill in the country, and that the product will be the same. This is not the case. Different ores, different sizes and shapes of ingots, different systems of rolling and treatment modify the product either for wear of the rails or their safety as girders.

Mills which make five or six rails per ingot, and cut the

blooms into two parts and roll in two and three lengths, produce a safer product as a girder for climates where the cold waves reduce the temperature to 20° or 30° below zero for a week at a time, than if they rolled the five and six lengths as a single bar and afterwards cut it into rail-lengths.

I made more than 600,000 tons of rails from 1890 to 1898 in my sections, principally the 65-, 70-, 75-, 80-, 95- and 100-lb. weights, and but few of these rails, after service of from 10 to 16 years, have developed split heads. It is probable that as the head becomes more worn, it may split, possibly due to a combination of a pipe and slag in the surface, or lateral splitting of the head from the slag in the bearing-surface, without a trace of a pipe in the web of the rail. Extensive investigations in the past two years upon slag, oxides and other impurities in the bearing-surface of the heads of the rails, leave no doubt as to their existence, and the causes of some types of rapid wear and failure of the rails as girders. Minute defects in ingot-structure, of little importance a decade since, are, under the present wheel-loads, the incipient sources of failure of the rail-heads.

The necessity of providing a range in the specifications for the chemical composition, so that it may be adjusted to local conditions, is imperative to produce sound ingots free from pipes and other defects, particularly as the run of the mines varies as to the traces of foreign elements in the ores from year to year.

I have made experiments in the casting of ingots with the large end at the top. The results were favorable, and confirm to a certain extent what has been said by Professor Howe. There are other advantages in casting with the large end up, in permitting more of the impurities to escape from the setting steel, without being caught in the corners of the columnar structure. The methods of teeming ingots in use at the present time are not suitable for this class of work. It would be necessary to reconstruct the plants for this purpose.

The most important feature which I have found essential for making sound ingots is, that more time should be allowed after recarburizing either in the converter or in the ladle before teeming, as brief agitation has not accomplished wholly the desired result. One of our most experienced steel-makers recently told me that he considered this to be a necessary condition of procedure for a reliable product.

Since this contribution was first written, several thousand tons of rails have been made by the Bessemer process, the steel being held in the converter 2 min. 30 sec. after recarburizing, with beneficial results—as formerly, when that was a part of the specifications. At another plant the steel was held 2 min. 30 sec. in the "intermediate" ladle. A time-element after recarburizing forms a part of the 1908 specifications of the New York Central Lines—a return to the practice inaugurated in 1891.

Professor Howe does not distinguish in the setting of the ingots between high-carbon and low-carbon steels. It is a well-known fact that steels of 10 or 15 points in carbon usually rise in the ingot and do not form pipes, and that it is necessary to cast such metal in bottle-mouthed molds and cap them to prevent the steel from overflowing. Increase the carbon, and the tendency to form a pipe from what is usually considered shrinkage of the metal becomes more pronounced, unless other methods are used to obviate it.

Teeming with a small-sized nozzle is also conducive to producing sound ingots and freedom from pipes and seams, and is a part of the 1908 specifications of the New York Central Lines.

The segregation of the ingots, while more or less associated with the piping of the upper part, may also be quite independent. The manufacture of my sections of rails according to my specifications has developed comparatively few pipes after years of service; the ingots were teemed 16 in. square, or less, on the base, and after stripping were thrown down into the pit, loaded on cars and charged into horizontal furnaces. The opportunity for the metalloids to segregate and rise to the top of the ingot is not so favorable in the horizontal furnaces as in ingots charged directly into vertical furnaces. The segregation in ingots for the high-carbon rails was not so marked as in ingots which at the present time are kept in a vertical position from the teeming to the blooming.

The general experience of teeming shows that in small ingots which set quickly the segregation is confined or limited to a marked extent. The size of the ingot must, however, depend in a degree upon the material which is to be subsequently produced. Large, flat ingots are essential for boiler-plate and structural work. Those of large dimensions are required for

armor-plate, also for large shafts for steam-ships. The size of the ingots for rails is also being increased, and the practical results from the 19- by 23-in. ingots of short length, compared with those from ingots 19 in. square and of longer length, are decidedly in favor of the larger stubby ingot.

The study of the pipe in ingots is usually made when the steel is cold and the entire shrinkage from hot to cold metal has occurred. I have recently had the opportunity of seeing some of the modern large-sized ingots for rails and other material cut through the center, and also the cores taken from the center of large ingots. The exhibits are apt to be misleading to any one not versed in the art of rail-manufacture, for it is difficult to conceive how such shrinkage-cavities could be overcome in actual practice, except by a greater discard than is usual. When the ingots were allowed to become cold before charging into the heating-furnaces, the cavity formed could only be removed by a large percentage of discard. But when the ingot is charged into the heating-furnaces as soon as stripped, the full shrinkage-cavity of the cold ingot does not develop, and in blooming, the lesser cavity may be pushed forward into and eliminated in the crop of the bloom by the usual discard. Several crops of good ingots have been cut, and this was found to be the case—as indicated by rails in service. The work which I did from 1890 until 1900, when cupola-metal was used and care taken in the composition, shows, by actual use in the track, that sound ingots have been made in which piping is the least of their defects. This can be and has been repeated in larger ingots with the advance in metallurgical knowledge of to-day.

I made 95-lb. high-carbon low-phosphorus rails from 14-in. ingots at the works of the Bethlehem Steel Co. for the Boston & Albany railroad in 1891 and 1892, and gave 5 or 6 min. in the ladle for the oxidation-products to escape from the chemical reactions of the recarburizer. The 75-lb. rails for Dr. Webb's Mohawk & Malone railroad were also made at the same time. These high-carbon low-phosphorus rails were rolled while John Fritz was general superintendent of the plant, for he was the only manufacturer who admitted at that time that such rails could be made. These were premium rails, and the late William Bliss, President of the Boston & Albany railroad, paid $2 per

ton additional for the high grade of steel, and Dr. Webb also paid the same premium for part of his rails. Mr. Bliss considered it economical to secure as good material as possible, and their service, most of them, after 16 years, being still in the track, has proved the wisdom of the initiatory procedure.

Mr. Fritz had constructed his 48-in. blooming-train for 16-in. and larger ingots prior to this time, but it cracked the skin of the ingots so much more than the former 26-in. blooming-train, that I had my choice of which size of ingots to use. After many tests, I chose the 14-in. ingot, which made two lengths of 30 ft. rails, while the 16-in. made three lengths.

The Boston & Albany rails, as also a part of the Mohawk & Malone rails, contained 0.60 of carbon and 0.06 per cent. of phosphorus. The heats, after recarburizing in the converter, were poured and remained in the ladle from 5 to 6 min. to allow the chemical reactions to take place and oxidation-products to escape before teeming. The ingots were stripped, thrown on their sides in the pits and charged into horizontal furnaces for equalizing the heat before blooming. The 26-in., 3-high blooming-train of 11 passes was used, and the blooms had a discard until sound steel was obtained, then chipped under the steam-hammer, again charged into horizontal furnaces for reheating and then rolled in 11 passes into rails. They were finished at between 950° and 1,000° C., as nearly as could be measured by the Siemens copper-ball and water pyrometer. We had not then advanced to the scientific requirements of sawing the rails 0.01 in. shorter for each second after leaving the rolls, and rolling frequently so cold as to damage the rail as a girder. Rolling the rail cold for wear is one factor to be considered, but this in any case must not exceed the safety-factor as a girder. The new type of sections with 0.5 in. thickness of edge can be rolled too cold for safety, as has occurred in some tests. The Bethlehem rails on the Boston & Albany railroad, after a service of 16 or 17 years, involving a large volume of traffic on heavy grades and sharp curvatures, have lost only about $\frac{1}{8}$ in. in height. They show by the sound inside corner of the head or the inside corner containing inclusions of slag, oxides, or gas whether the rails were from the bottom or the top of the ingot. The fractures of these rails, on both the Boston & Albany and the Mohawk & Malone railroads,

have been exceedingly rare after so many years of service. The rails have excellent wearing-properties and are also tough as girders.

More than 600,000 tons of rails with carbon 0.60 and phosphorus 0.06 per cent. were rolled at Scranton, Pa., in my sections, from ingots 14 to 16 in. sq. on the base, of sufficient length for three 30-ft. rails. Special effort was made to secure a composition which would set well in the ingots and be free from pipes. The number of stickers broken under the drop was sufficient to show the general soundness of the ingots.

Commencing December, 1892, the rails from the ingot were marked *A*, *B*, and *C*: the upper rail, *A;* the middle, *B;* and the lower, *C*. These letters can be found upon the rails and the wear of each easily traced. But few of them have pipes in the web, and only a few split heads have occurred. The ingots were thrown down in the pit, loaded on cars and charged into horizontal furnaces for equalizing the heat, and then were rolled direct without reheating. The *A* rails show that occasionally there were entrained slag, gas, and other impurities at the junctions of the columnar structure of the small-cornered molds. The *B* and *C* rails show after many years' service hardly any breaking down of the inside running-edge of the heads, upon either curves or tangents. The fractures of these rails in the track, even under heavy traffic, have been slight, and but few more have occurred in the *A* than in the *B* or *C* rails. Some of the upper ends of the *A* rail have been crushed or have worn slightly faster than the other portions under heavy traffic. The corner radius of the ingots was about 2 in., which was too small, and metal would set and cool faster than in the sides of the ingot, while some blow-holes would form in the upper corners of the ingots.

It is noticed in taking sections of the different lettered rails, that in the *A* rail the impurities are largest in amount, as would be expected. There is often a well-defined central core, which is not as readily seen in the *B* rail, and only rarely noticed in the *C* rail.

Professor Howe, in his paper, says that the matter of noting the different rails or blooms from the upper part of the ingot has been until now entirely overlooked. I inaugurated the practice in December, 1892, of marking the rails rolled as fol-

lows: the rail from the upper part of the ingot, *A;* from the middle, *B;* and from the lower, *C.* In case of four or five rails I have used the letters *D* and *E.* This practice was never carried out at the Carnegie mills, but it was by the Lackawanna Iron & Steel Co., at Scranton, and also at their Buffalo mill. The practice of designating the different rails from the ingot is now becoming quite general, and will soon be required of every mill in the country.

The defects of ingot-structure of the rails made from the 14-in. ingots, and subsequently in 1894 from 16-in. ingots, are not due so much to piping and segregation as to the collection of the impurities in the upper (*A*) rail. When, in 1898, the ingots were charged into vertical furnaces, to equalize the heat, the results of segregation became more pronounced. In the horizontal furnaces, the metalloids separating from the molten metal, would rise towards the upper side of the ingot and be harmless in most cases, while in the vertical furnaces they rise and concentrate in greatest amount in the upper part of the *B* or the lower part of the *A* rail.

When the ingots were charged in the horizontal furnaces those of high carbon-content did not segregate sufficiently to be of serious consequence, as is shown by the wear or fracture of the rails.

In rails rolled from the present large ingots, 19 in. square upon the base and from 5.0 to 5.5 ft. long, the segregation becomes decided, and often included slag is found in the bearing-surface of the head, which spreads laterally under the moving wheels and produces a split head, while there is scarcely a trace of pipe in the web.

Rails made from large ingots, 20 by 23 in. on the base and short in comparison to the width, show fewer split heads from segregation and a smaller quantity of entrained slag in the head of the rail than those made from the smaller ingots of greater length. There is one brand of rails rolled from the comparatively small but long ingot, in which the split heads after five or six years of service amount to 8 or 9 per cent. of the entire product for July, August or September rollings.

The shortening of the time in the Bessemer department from recarburizing to teeming has contributed to the inferior quality of much of the large outputs of Bessemer rails. The segre-

gation in the long ingots, particularly when the phosphorus is at 0.10 per cent., is a serious matter with our present service of high-speed trains.

Observing the different brands of rails, containing 0.10 per cent. of phosphorus, regardless of the carbon-content, the wear is not as satisfactory as from the high-carbon and low-phosphorus rails. The fact of the large amount of slag which is found in the head of the rails is now well established in different brands and its composition is becoming known.

The corners of the molds at one mill, I had made of larger radii, and this has helped to some extent to relieve the columnar structure in the corners of as much entrained slag and impurities as was formerly found in the corners of smaller radii, but the shape of the mold alone does not permit of the escape of an unduly large amount of slag and impurities under rapid teeming. It is the requirement of sufficient time, and the Bessemer departments should not be run too rapidly, in order that each converter or ladle should be given ample time for the chemical reactions and oxidation-products to escape. This arrangement requires more ladle-capacity to give sufficient time for each operation, which of itself would not necessarily reduce the output of the mill when properly planned.

The high phosphorus-content is a serious question, and unless some relief is quickly found it may be necessary to use the basic open-hearth process for reducing this content of phosphorus. Steel made by the basic open-hearth process, in which proper time is allowed for the escape of the oxidation-products, has proved exceptionally tough and free from included slag. It gives evidence of being well adapted not only for a slow rate of wear, but also for safety in the sections as a girder. The heavy bases in the new design of rails are so massive that the time required for the distribution of strains makes them more sensitive to rapid strains than the lighter bases of some of the present rails.

These questions will require time to investigate thoroughly and find out how to roll properly the sections for the slowest rate of wear and the greatest safety as girders.

The cutting of the lettered rails into sections and examining the segregation, shows that for the horizontal furnaces those effects were not as marked as those from the larger in-

gots in the vertical-fired furnaces. The ingot-structure as a manufactured product is also shown, and it is a good way to investigate and study the subject. It now seems almost safe to say, from present studies, that the ingot-structure will improve more in the coming year than it has done in the past decade.

PROFESSOR HOWE:—I think that Dr. P. H. Dudley has put his hand on the thing which we have hitherto overlooked—namely, the opportunity for the oxide and slags to be removed from steel. I think the late experiences with electric furnaces show that he is right, if you consider what the electric furnace does. Of course, the electricity as electricity has nothing to do with it, but what it does is to allow the metal to stay stationary for a long while. The Heroult process removes much sulphur also, but that by itself does not suffice to explain the improvement. It seems to me the great improvement is due to the ample time for the gradual removal of the inclosed slag.

HIRAM W. HIXON, Philadelphia, Pa.:—Concerning the segregation of ingots in copper-converting and silver-refining, in these processes a large quantity of gas is given off by the metals at the time of solidification. In the case of silver, oxygen absorbed from the air during the process of smelting is given off with such force at the time of solidification that it shoots out sprouts, proportionate to the size of the bar or button that is solidifying. Copper as finished in a converter has the property of holding SO_2 gas in solution until it solidifies. I have frequently observed, in turning down a converter, after the charge has been finished properly and is in a quiescent condition, that as soon as it begins to chill a little it will begin to boil, giving off SO_2, and, when it is poured into molds, this action is very much accelerated and the gas given off is of greater strength and is very irritating to the nostrils and the eyes. If the charge is poured into a ladle, and that ladle-ful of copper poured into a furnace, the copper can be held in a molten condition until the gas is given off, and then comparatively clean castings of copper can be made from the same copper that would have given very rough bars if cast direct. I

have often seen little copper-volcanoes at work on the top of a copper bar, and sometimes they are really dangerous. In the Bessemer process of making steel a very large quantity of air is blown through the steel or through the cast-iron being converted into steel, and there is more or less similarity in the absorption of gases in this process and in the case of silver and copper. It is undoubtedly true that segregation of the gases towards the top of the ingots and the good rails coming from the bottom, is due more to the occluded gases than to any other cause. Therefore, the open-hearth products make better rails, simply because the steel is held for a long time in a molten state and every opportunity is given for the occluded gases to escape.

That point is made clear also by the fact that steel made by an electric smelting-process is much superior to steel made by any other process, even though its analysis is similar, because there is no included gas in it. Therefore, if you want to improve the ingots of the Bessemer process the opportunity is offered, instead of casting direct from the ladle, to put that steel into a large reservoir or furnace for that purpose and hold it for a considerable time, giving an opportunity for the included gases to escape. Segregation will be found to be much less after the steel has given off its gas than it will if cast immediately from the ladle.

A. A. Stevenson, Burnham, Pa.:—Referring to Professor Howe's able and exhaustive paper on the Piping and Segregation of Steel Ingots, the theories advanced and the facts brought forth are extremely interesting. Professor Howe reaches a number of conclusions based on certain data and some rather close reasoning. How far these conclusions agree with results in actual practice, is a natural question. In the main, as far as our own experience is concerned, the facts agree with the conclusions of Professor Howe.

In the beginning, the following remarks, unless otherwise stated, refer to high-carbon acid open-hearth steel, such as is used for tires, of the following approximate analysis:

	Per Cent.
Carbon,	0.60 to 0.70
Silicon,	0.20 to 0.30
Manganese,	0.60 to 0.80

The ingots vary in weight from 1,500 to 6,000 lb., and are bottom-cast in groups.

As far as the supposed genesis of a pipe in a solidifying steel ingot is concerned, the illustrations given in Professor Howe's paper do not, in our opinion, give the correct shape, although the description in the text does agree with the actual shape in-

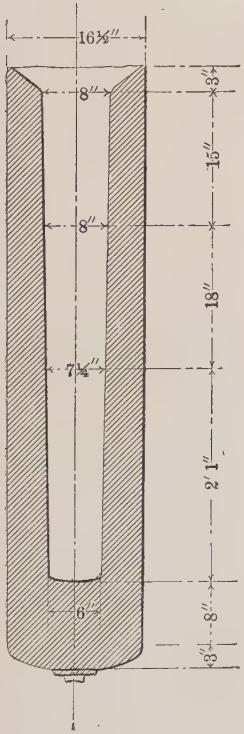


Fig. 1.—Section of 16-in. Ingot Tipped and Emptied 15 Min. After Pouring.

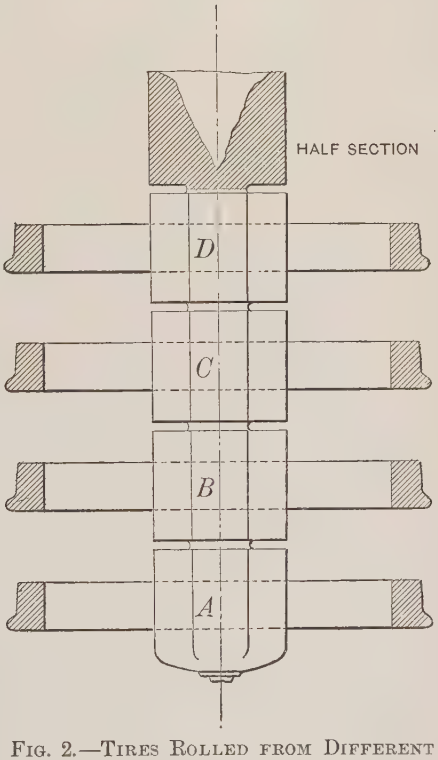


Fig. 2.—Tires Rolled from Different Portions of Ingot.

dicated by Fig. 1, which shows a section of a 16-in. bottom-poured ingot tipped over 15 min. after it was poured and the fluid portion of the steel emptied out. Fig. 1 shows the rapidity with which an ingot of this size solidifies.

With regard to the question of increased tendency to segregate with increased size of ingot, Professor Howe, in his discussion to-day, brings out the point, as a result of experiments made since his original paper was presented, that apparently

there is not a greater tendency to segregate with an increase in size of ingot.

In the paper referred to by Professor Howe,[1] which was presented by Mr. Kent and myself at the Chicago meeting in 1893, we brought out the fact of the large amount of segregation that sometimes occurs in small ingots, and gave several examples. One of the most pronounced cases of segregation was in an ingot of about 400 lb. weight, and the worst points of segregation did not seem to be at the bottom of the pipe, but along the sides. In this ingot the segregation was so pronounced that it showed plainly on etching, and was readily distinguishable from the rest of the metal by the porous appearance, apparently due to the greater ease of attack by etching-acid. The segregation appeared in pockets or globules, and the line of demarcation between the segregated portion of the ingot and other portions was very pronounced.

In a number of other cases, in ingots varying from 400 to 800 lb. in weight, we have found segregated spots in which one or all three of the elements—carbon, phosphorus, and sulphur—have shown from 50 to 100 per cent. more than in the balance of the ingot. No doubt the segregation was even greater, but there was no way of getting drillings that did not contain some of the metal surrounding the porous or segregated spots. The ingots in question were cast in iron molds, and consequently must have cooled very quickly, which, according to all reasoning, would tend to decrease the segregation, and the question arises whether or not, in these cases, the concentration of the elements was not due to an extrusion or squeezing-out process, owing to the rapid freezing.

It may be interesting to know that in tires made from short ingots the segregation was as readily detected by the etching-process in the tire as in the ingot itself. Since our paper was read, in 1893, we have made a great many tests of larger ingots and tires from larger ingots. We have found that segregation in ingots as high as 6,000 lb. in weight, even around the pipe, was comparatively small, but in no case was the segregation as extensive as in the small ingots. In all our experiments with long ingots and tires made from them we have never found segregation so pronounced as to show porous spots and

[1] *Trans.*, xxiii., 637 (1893).

streaks by etching. This, in itself, tends to show that the segregation is not altogether an element of size. Our opinion is that the condition of the bath at the time the heat is poured has a great deal to do with the amount of segregation, and that the careful manipulation of the bath in the furnace is as important as the size or shape of the ingot, if not more so.

Some years ago we installed 50-ton furnaces for making tire-steel. It was a question whether or not the steel in heats of this size could be made as homogeneous as the steel in smaller heats, which was the then-prevailing practice in the manufacture of tire-steel. In order to settle this point we made several hundred tests by taking tires from the bottom or *A* billet, as shown in Fig. 2, and analyzing borings and turnings from these tires; then taking the tires from the top or *D* billet of same ingot and analyzing borings and turnings from same. It is understood, of course, that in the manufacture of tires there is a small plug punched out which might contain segregation, as it is from the exact center of the ingot. In practically all cases we found no greater variation in the analyses of the borings and turnings from the *A* and the *D* tires from the same ingot than would be represented by variation in chemical manipulation. In a few cases the analysis of the borings from the top or *D* tire showed slightly higher results, but in no case was this difference sufficient to cause any disquietude. The *D* tire is just under the pipe, and if there were any degree of segregation below the pipe we think that it would have been detected, in view of the large number of tests that were made.

In addition to these tests, a great many tires were cut up and analyzed at various points to see if the uniformity extended throughout the mass, and never was any greater difference found than might be accounted for by allowable chemical variation. Many ingots were also cut up, and, in the immediate vicinity of the pipe, there were never found such extreme cases of segregation as occurred in small ingots.

Fig. 3 shows the section of a 20-in. ingot which was laid off in 2-in. squares, and drillings taken from the intersections for analyses. Table I. gives the results from various drillings, which show a slight segregation in the vicinity of the pipe and along the axis of the ingot. In the main, the analyses show great uniformity.

As far as the mode of solidification is concerned, we find that in the grade of steel used for tires the solidification is invariably not of the "onion-skin" type, but of the "land-locking" type, which proceeds by sending out large pine-tree crystals. We have found this to be the case in medium steel such as is used for locomotive forgings, in soft steel, and in high-carbon crucible-steel, and we believe it is the usual mode of solidification.

In the discussion of rail-steel and ingots from which rails are made, much has been said concerning columnar structure, but we have never yet found a case in our own practice.

Streaks, such as are frequently referred to in the discussion of steel for government work, and which at the present time seem to be giving the government considerable uneasiness, we do not have.

Referring to Professor Howe's remarks with regard to the bridge in the pipe of an ingot, we think that this is due, in many cases, not so much to the fact that the steel at the top of the ingot freezes to the point where it is self-supporting and the metal sinks away as the ingot cools, as to the fact that support is given to the top shell by escaping gas. This is especially true of the bridge over the top of the pipe. The top surface chills and in a sense seals the gas that is escaping from the steel, and this gas, rising up through the metal, acts as a support before the shell itself becomes self-supporting. We are confirmed in this belief by the fact that in many cases the under side of the bridge is practically smooth, and has no crystals hanging to it. This effect is shown in Fig. 4.

Referring to Professor Howe's recapitulation, the various points mentioned are discussed in his order:

"The pipe is shortened, though probably at the cost of increasing the degree of segregation:"

1. *By casting in wide ingots.* We understand from this that Professor Howe thinks that casting in larger ingots would tend to decrease the length of the pipe. As far as our own steel is concerned, we do not find that this is the case. The depth of the pipe seems to increase with the size of the ingot, and as far as the depth of the pipe is concerned, it is practically the same in ingots of the same diameter, irrespective of the length. We are speaking now of a visible pipe. Fig. 5 shows a photograph

TABLE I.—*Analyses of Drillings from 20-in. Ingots.*

Hole.	Comb. C. Per Cent.	Si. Per Cent.	P. Per Cent.	Mn. Per Cent.	S. Per Cent.	Hole.	Comb. C. Per Cent.	Si. Per Cent.	P. Per Cent.	Mn. Per Cent.	S. Per Cent.
1	0.660	0.280	0.048	0.68	0.039	94	0.630	0.275	0.058	0.67	0.048
3	0.634	0.270	0.046	0.68	0.040	95	0.630	0.265	0.059	0.67	0.047
7	0.660	0.270	0.052	0.66	0.041	96	0.650	0.270	0.052	0.65	0.046
10	0.640	0.280	0.055	0.68	0.047	99	0.660	0.260	0.054	0.68	0.043
11	0.645	0.280	0.052	0.65	0.038	101	0.645	0.275	0.054	0.67	0.041
14	0.645	0.280	0.054	0.66	0.042	103	0.645	0.280	0.055	0.65	0.048
17	0.620	0.275	0.050	0.66	0.039	104	0.645	0.260	0.052	0.68	0.047
18	0.630	0.280	0.055	0.67	0.043	105	0.640	0.270	0.057	0.65	0.046
19	0.654	0.280	0.055	0.68	0.045	107	0.670	0.275	0.054	0.67	0.043
20	0.645	0.280	0.054	0.68	0.043	111	0.669	0.280	0.053	0.66	0.044
21	0.660	0.280	0.052	0.67	0.041	112	0.645	0.280	0 056	0.67	0.046
25	0.660	0.280	0.052	0.67	0.043	113	0.645	0.265	0.053	0.66	0.047
26	0.660	0.280	0.052	0.68	0.043	114	0.660	0.280	0.053	0.68	0.047
27	0.640	0.275	0.054	0.66	0.043	115	0.650	0.270	0.054	0.69	0.045
29	0.660	0.270	0.052	0.67	0.045	121	0.660	0.280	0.056	0.67	0.050
30	0.645	0.280	0.055	0.66	0.044	122	0.641	0.260	0.053	0.66	0.048
34	0.645	0.280	0.056	0.68	0.044	123	0.651	0.275	0.055	0.67	0.048
35	0.660	0.275	0.056	0.66	0.044	130	0.654	0.280	0.053	0.67	0.044
36	0.660	0.275	0.055	0.66	0.043	131	0.640	0.280	0.057	0.68	0.041
39	0.654	0.270	0.055	0.67	0.047	132	0.666	0.280	0.055	0.69	0.045
40	0.660	0.280	0.053	0.68	0.052	140	0.690	0.275	0.054	0.67	0.039
42	0.693	0.285	0.057	0.68	0.052	149	0.670	0.275	0.053	0.67	0.042
43	0.645	0.280	0.053	0.67	0.047	156	0.646	0.265	0.052	0.68	0.046
46	0.660	0.278	0.056	0.66	0.047	158	0.650	0.275	0.052	0.66	0.046
48	0.654	0.280	0.052	0.67	0.046	160	0.645	0.275	0.054	0.66	0.047
49	0.660	0.275	0.054	0.65	0.046	172	0.660	0.280	0.054	0.67	0.046
51	0.660	0.280	0.054	0.68	0.052	176	0.654	0.275	0.053	0.66	0.041
52	0.645	0.280	0.054	0.65	0.050	180	0.660	0.260	0.054	0.67	0.043
54	0.645	0.270	0.055	0.65	0.045	191	0.642	0.275	0.053	0.68	0.042
57	0.660	0.280	0.053	0.67	0.046	194	0.645	0.270	0.051	0.68	0.040
58	0.660	0.275	0.054	0.67	0.046	197	0.645	0.270	0.053	0.68	0.045
60	0.660	0.280	0.052	0.66	0.050	199	0.630	0.260	0.054	0.66	0.042
61	0.645	0.265	0.053	0.65	0.043	203	0.640	0.270	0.055	0.66	0.044
66	0.654	0.280	0.055	0.66	0.045	207	0.660	0.280	0.054	0.67	0.041
67	0.680	0.280	0.053	0.65	0.049	221	0.645	0.260	0.051	0.66	0.042
69	0.660	0.280	0.052	0.65	0.046	230	0.630	0.270	0.053	0.65	0.040
70	0.660	0.280	0.055	0.68	0.044	253	0.642	0.275	0.053	0.66	0.042
75	0.645	0.270	0.054	0.65	0.047	255	0.645	0.280	0.055	0.68	0.041
76	0.645	0.280	0.054	0.68	0.048	257	0.648	0.275	0.053	0.68	0.039
78	0.675	0.270	0.052	0.64	0.046	259	0.651	0.280	0.052	0.67	0.042
79	0.654	0.265	0.053	0.66	0.047	261	0.648	0.280	0.055	0.66	0.043
84	0.630	0.270	0.053	0.67	0.044	280	0.660	0.275	0.053	0.66	0.041
85	0.645	0.275	0.054	0.67	0.042	284	0.642	0.260	0.052	0.65	0.041
86	0.690	0.275	0.058	0.66	0.048	288	0.645	0.260	0.045	0.68	0.041
87	0.675	0.265	0.054	0.67	0.047	298½	0.630	0.280	0.052	0.67	0.043
88	0.660	0.280	0.052	0.68	0.048	302	0.660	0.270	0.053	0.65	0.043
91	0.675	0.270	0.055	0.66	0.045	305½	0.645	0.275	0.053	0.64	0.040

of the section of a 10-in. and a 20-in. ingot, cast in the same group. The ingots were bottom-cast. It is true that the depth of the pipe in the 10-in. ingot is less than in the 20-in. ingot. As far as the center of the ingot below the pipe is concerned, which shows apparently solid when planed, etching shows a slight tearing apart, which we think is due to contraction-strains on the mushy center before solidifying, as suggested by Professor Howe.

2. *By casting in molds of low conducting-power—i. e., lined with sand or clay—especially if pre-heated.* In a few cases we have made octagonal ingots in sand molds of the same size that we make in our regular practice in chilled molds. In each case the piping in the ingot cast in the sand mold was at least twice as deep as in the same sized ingot cast in the chilled mold. It may be that there is a greater virtual expansion in an ingot cast in a sand mold, owing to the fact that original cooling is not so rapid as in an iron mold, and this might account for the greater depth of the pipe.

"The pipe is shortened and the segregate raised:"

3. *By top-casting instead of bottom-casting.* In our practice we have found very little difference in the length of pipe whether the ingot is top- or bottom-cast. Our regular practice is to bottom-cast, but we have made a few top-cast ingots for experimental purposes. In the top-cast ingots which we made we found no appreciable difference in the pipe, and in the grade of steel we use there are practical reasons why bottom-casting is preferable to top-casting, and any slight difference that we might find in the length of the pipe would be more than counterbalanced by other points in favor of bottom-casting. Fig. 6 shows two small ingots, each weighing about 615 lb., which were cast in iron molds. *A* was top-poured and *B* bottom-poured. The ingots were not cast in the same heat, but the analyses given below correspond very closely:

	Combined C.	Si.	Phos.	Mn.	S.
A, . . .	0.675	0.280	0.044	0.68	0.050
B, . . .	0.645	0.285	0.045	0.63	0.053

It will be noticed that, as far as the depth of the pipe is concerned, there is very little difference in favor of the top-poured ingot, and this is doubtless due to the small sink-head, which was not used in bottom-pouring,

4. *By slow casting.* We believe Professor Howe's reasoning with regard to slow casting is correct. In our practice the large groups help counterbalance the necessity of rather rapid pouring; assuming that we start pouring at the proper temperature, the steel should be got into the molds as rapidly as possible.

5. *By casting with the large end up instead of down.* Regarding the commercial feature of casting the ingot with the large end

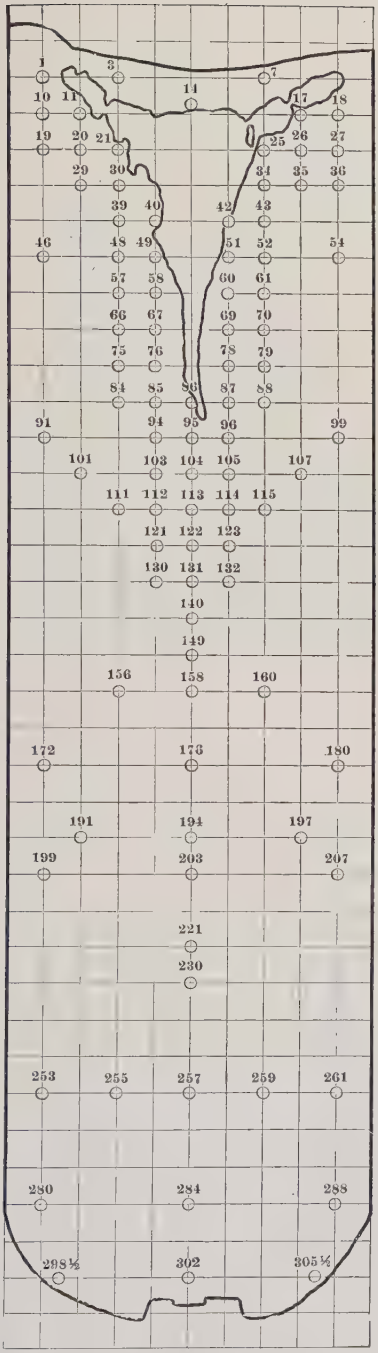


Fig. 3.—Points from Which Drillings Were Taken for Analyses of Table I.

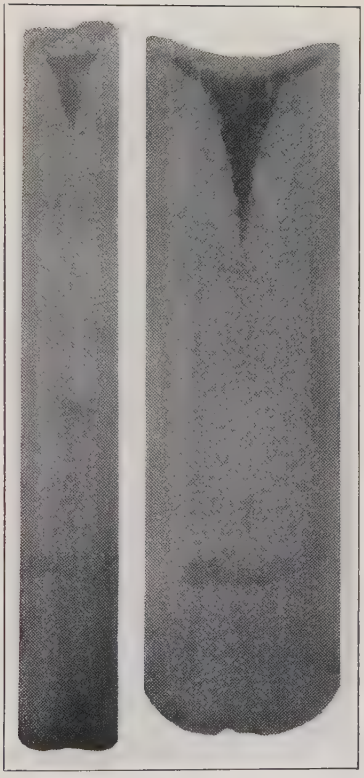


Fig. 5.—Sections of a 10-in. and a 20-in. Ingot, Cast in the Same Group, Showing Effect of Size of Ingot on Depth of Pipe.

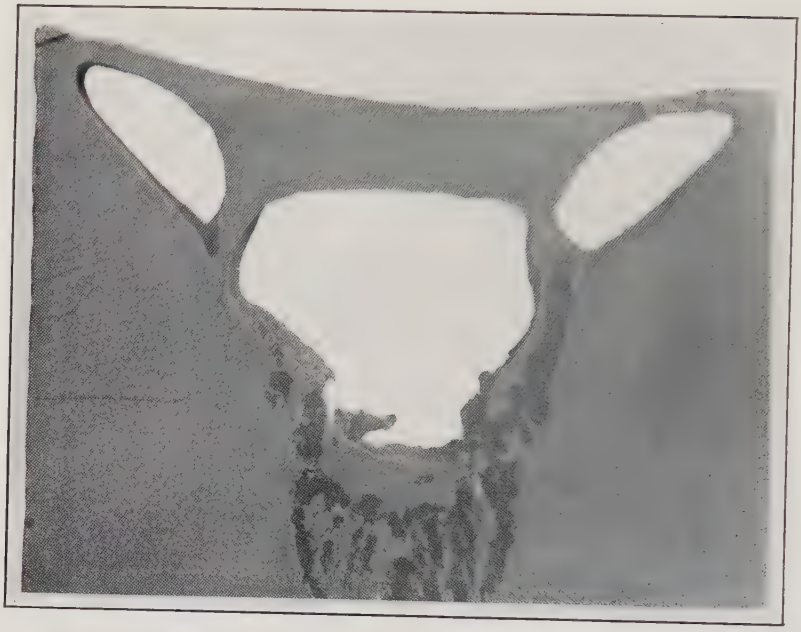
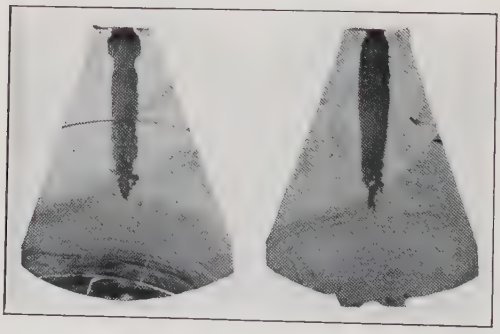


FIG. 4.—BRIDGES WITH SMOOTH UNDER-SIDE.



A, top-poured. *B*, bottom-poured.

FIG. 6.—SHOWING EFFECT OF TOP-POURING AND BOTTOM-POURING ON DEPTH OF PIPE.

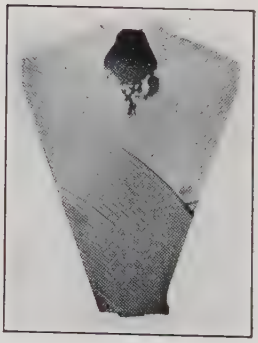


FIG. 8.—DEPTH OF PIPE DECREASED BY CASTING WITH LARGE END UP.

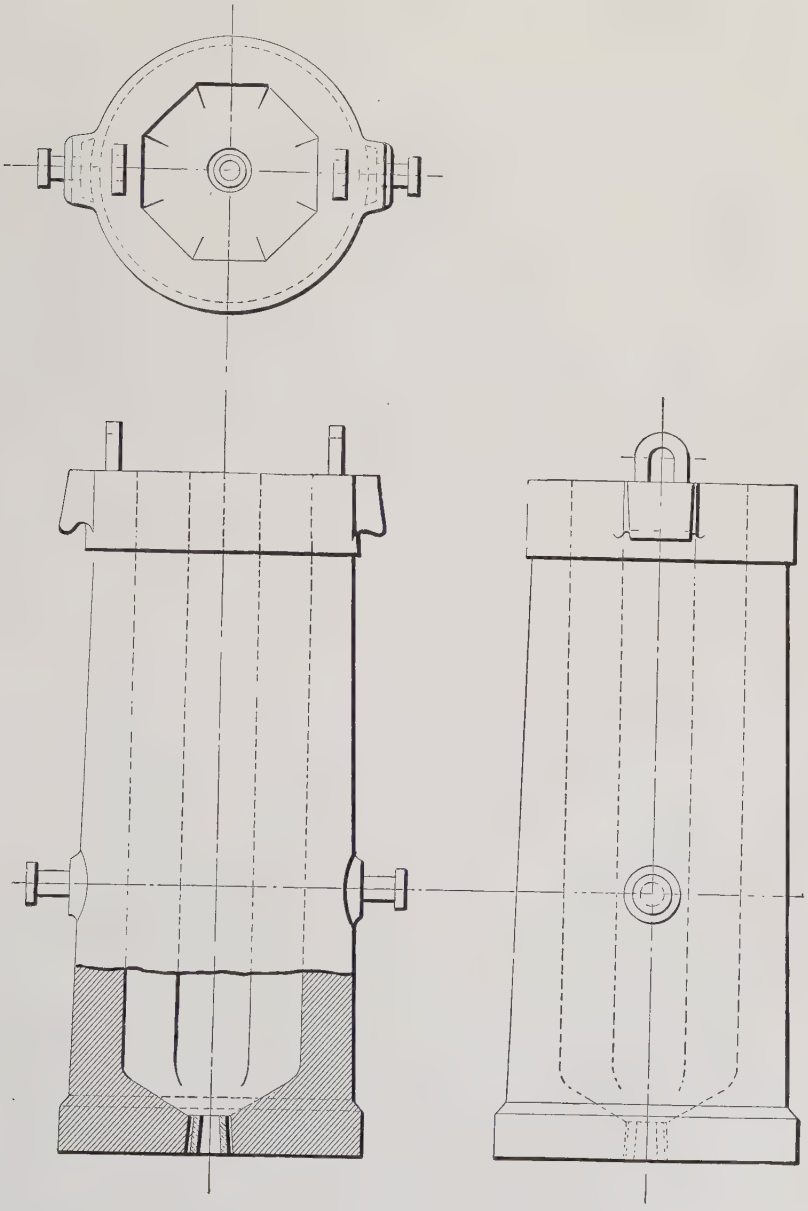


FIG. 7.—PLAN, PART SECTION, AND ELEVATION OF MOLD.

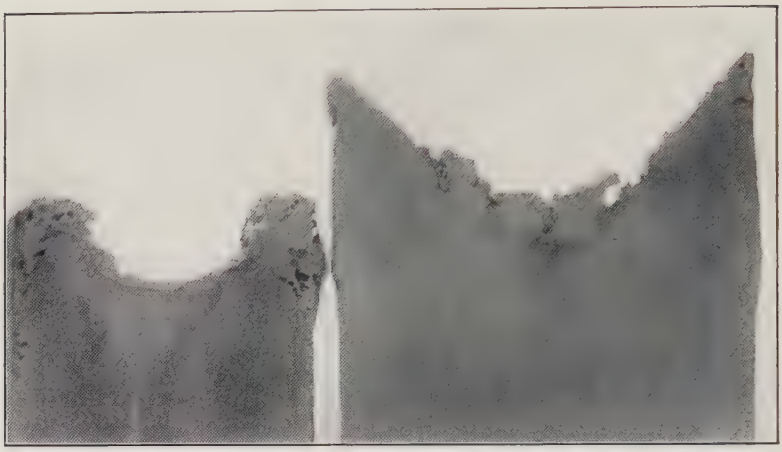


FIG. 9.—SECTIONS OF INGOTS, SHOWING THE EFFECT OF RETARDING THE COOLING OF THE TOP BY MEANS OF PLUMBAGO.

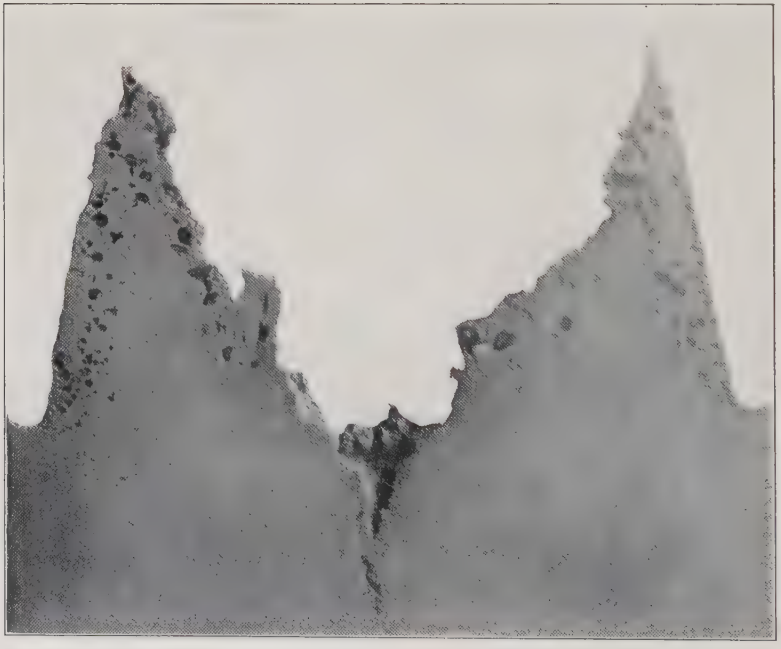


FIG. 10.—SECTION OF INGOT CAST IN A MOLD WITH A SAND LINING AROUND THE TOP, AND USING PLUMBAGO.

up, this has been our practice for the past 15 years. Our molds have solid bottoms, as shown in Fig. 7. We have been casting the output of three 50-ton furnaces in this type of mold, and have found no difficulty in operating. So little trouble has arisen that the plant has no ingot-extractor; the ingots are emptied out, and there is very rarely a sticker. This method of casting ingots was not adopted on account of any benefit that might be derived from shortening the pipe, but for other reasons. We have, however, made some bottom-cast ingots in molds with open bottoms, and these ingots are, of course, slightly smaller at the top than at the bottom. There was no commercial difference in the length of the pipe; in fact, if there was a difference it was not sufficient to attract attention. The large end of the mold in each case was about 0.75 in. larger in diameter than the small end. The question arises as to the amount of variation in diameter between the large end and the small end, which would make an appreciable difference in the pipe.

Fig. 6 (*B*) shows an ingot that was bottom-poured with small end up. Fig. 8 shows an ingot of the same dimensions cast in an iron mold, bottom-poured with large end up. In the latter case there is a very appreciable difference in the length of the pipe; the ingot cast with the large end up showing a much more shallow pipe than that cast with the small end up. These ingots were not from the same heat, but the analyses are almost identical.

6. *By retarding the cooling of the top by means of a sinking-head or otherwise.* As far as the shortening of the pipe is concerned, retarding of the cooling of the top of the ingot has a decided effect. As for segregation, we have not been able to detect much difference. Fig. 9 shows a section through the pipe of two ingots. Plumbago was put on the top of these ingots when the steel was within a short distance of the top of the mold.

Fig. 10 shows a section of the top of an ingot cast in a mold having a sand lining around the top of the mold, and using plumbago also.

7. *By permitting deep-seated blow-holes to form through adjusting the quantity of silicon and manganese or their equivalent.* In accordance with Brinell's experiments, blow-holes are not ex-

pected in steel of the analysis under discussion. We have, however, made tires to specifications that call for analysis that would permit the deep-seated blow-holes, but we cannot agree with Professor Howe that this would be a good method of shortening the pipe. We certainly prefer the pipe to deep-seated blow-holes. We do not believe there can be any thorough welding up of blow-holes in this grade of steel; we know that in many cases there is not. This statement also applies to steel of the grade used for locomotive forgings. In the case of low steel, such as is used for plates, there are many cases of blow-holes that are not welded. With blow-holes in tire-steel the tires naturally would wear shelly in service, and shelliness of tires seems to be the *bête noire* of the tire-manufacturer. The prevailing practice 15 or 20 years ago was to make tires from short ingots, but a shelliness was developed due to piping and gas-holes; this source of shelliness, however, has been avoided by cropping the top or pipe portion of the ingot.

The tire-manufacturers of to-day are again facing the problem of shelliness. An extensive series of investigations, covering a period of several years, the results of which were contained in a paper read by G. L. Norris before the Western Railway Club in Chicago last October,[1] would indicate that this shelliness is largely due to conditions of service, combined with increased wheel-load, rather than to any inherent defects in the steel.

8. *By liquid compression.* We have had no actual experience with liquid compression.

"The degree of segregation is lessened:"

9. *By quieting the steel by adding aluminum or its equivalent.* There seems to be no question as to the degree of segregation being lessened by adding aluminum or its equivalent, but we desire to add to what Professor Howe has said, that the amount of segregation also depends, in a large measure, on the condition of the bath and slag immediately before tapping.

10. *Probably by casting in small instead of in large ingots.* We believe that Professor Howe has changed his opinion somewhat as to this point, and, as stated previously, this does not agree with results obtained by us.

11. *Casting at as low a temperature as practicable.*

[1] Steel Tires—Causes of Defects and Failures, *Proceedings of the Western Railway Club*, vol. xx., No. 2, p. 20 (Oct. 15, 1907).

13. *Casting slowly.* Would it not be better to say, casting at as low a temperature as advisable? Experience would tend to show that it is not advisable to cast at as low a temperature as is practicable. Admitting that casting at a low temperature and casting slowly are advisable, in practice the conditions, in a sense, oppose one another. We refer particularly to the casting of large heats. Starting at the proper temperature, the heat must be poured rapidly, and by group-casting we think we approach as nearly as practicable the two conditions.

12. *Casting in thick-walled iron molds.* That the degree of segregation may be lessened by casting in thick-walled iron molds seems a reasonable conclusion, which our experience with long ingots apparently confirms. At the same time, it does not necessarily follow that the segregation will be lessened by casting in thick-walled iron molds, since small tire-ingots cast in such molds frequently segregate badly.

Prof. William Campbell, New York, N. Y.:—I have made a few notes on some of the things that the previous speakers have said. The first is about the shelling of tires. Some years ago there were submitted to me specimens of six tires of different manufacture, and I was asked to find out why they shelled, by the microscope. I cut out sections, and found in every case that the reason for the shelling was a very small film of oxide or parting, due to blow-holes.

Mr. Kinkead remarked that they had taken some 0.40-carbon steel and by annealing had produced the same results as obtained in the Bethlehem tests on nickel-steel. Since Professor Howe has been at Columbia we have been carrying out experiments on the annealing, etc., of steel all the way from 0.04 up to 2 per cent. of carbon, and we often find that we get really wonderful results. The temperature-limits are very small indeed. For instance, with a steel of, say, 0.50 carbon and 1 per cent. of manganese, the limit within which these results of strength with high ductility can be produced is about 50°, and seems to lie between the two critical points of heating: AC_1, a little above 700°, and AC_2, a little above 750°. That is the first factor. The second factor is the time. The time-element in annealing castings is very important indeed. We have some experiments under way under Professor Howe's direction that show that ordinary heating and cooling of a piece $\frac{3}{8}$ in. square will not

give refinement, but that it takes in some cases two or three hours for such a small piece.

Mr. Huston suggested that we divide steel into mild, medium, and high, as well as into Bessemer, open-hearth, etc. Why not go a step further and grade it according to the analysis? The analysis of the whole steel series gives us a basis of comparison, whereas if we say that we are dealing with a piece of mild steel the manufacturers of tubes, boiler-plate, etc., understand one thing, whereas the structural engineer understands quite another thing. Recently I obtained what I was told was mild steel, and, under the microscope, it looked to be 0.40 carbon, and the analysis showed that it was. Yet the manufacturer that made that steel classified it as mild steel.

It has been said that work at a high temperature does not affect the structure. This is a fallacy that has been accepted for a long time. Many of us have the idea that the finishing-temperature is the main factor determining structure. We had the idea that we were finishing our steel rails at near the critical point. For the last two summers I have been working at this subject with pyrometers, and I have found that these temperatures for finishing vary all the way from 950° to 1,100° C. When I so reported I was laughed at, but the pyrometer gives these readings, and when I asked one of the inspectors how he got his temperature, he said, "The rail runs down to the cutting-saw and there a piece of about a foot long is cut from the waste end, a workman puts it in a bucket of water and we read the rise in the temperature of the water, and I am finishing at 850°." With that kind of a pyrometer you can get a great deal lower than 850°. I do not think there is any question that the finishing-temperature is not the prime factor determining grain-size. I think the amount of reduction in the rolls has a great deal to do with the final product.

We have heard that in the case of faulty material, such as bad tires and rails, chemical and microscopical analyses indicate nothing. I agree that the former alone is often useless, but the two together, in nine cases out of ten, will solve the difficulty.

Again, it has been said that if you take boiler-plate and pull a piece that has been cut longitudinally, and another piece that

has been cut across, you will get different results; also that iron pulled across the grain breaks more easily; as if the grain of the iron had anything to do with it. It is not the grain, it is the slag. The grains are all the same. With good wrought-iron the grains measure the same, whether you take them across or through; but it is the slag-lines that give you the weakness with pieces cut across. In the same way with low-carbon steel, on cutting a section parallel to the direction of rolling, the structure is often found to be decidedly banded. The ferrite and pearlite bands are quite distinct, due to segregation, and etching shows up light and dark streaks. In the ferrite bands—*i.e.*, those poor in carbon—we find much or most of the manganese sulphide and silicates, drawn out in elongated masses and threads, and here also the phosphorus segregates. Hence, a section parallel to the direction of rolling will prove much stronger and more ductile than one cut across. A similar state of affairs is found in rail-steel, the effects of manganese sulphide and silicates being even more marked at times.

With regard to the two different types of ingot-structure, there is a polished ingot at Columbia which is an excellent example. For a depth of 0.75 in. the metal is composed of comparatively small regular grains, beyond which it is built up of long pine-tree crystals perpendicular to the sides of the mold.

Henry D. Hibbard, Plainfield, N. J. (communication to the Secretary*):—These remarks refer, except when stated otherwise, only to the harder grades of steel, and not to what is known as soft steel, in which the only thing which classifies it under the name of steel is the fact that it is made by a fusion-process.

These harder steels are or should be made by what I have termed the "solution" process of steel-making, in which the gases are kept in solution during solidification as completely as practicable, as distinguished from the "evolution" process practiced in making soft steels, in which the gases are encouraged to evolve freely from the metal while it is freezing.

Many of the points for procedure in the manufacture of steel which are laid down and emphasized have for their object the cure of ills which should not exist, and such points are likely to

* Received Mar. 3, 1908.

be considered especially essential by some one who has found them or some of them to have benefited steel made by his established methods.

Broadly speaking, the better the steel is made the more it will pipe. Whether or not proper treatment before casting affects or reduces the central segregation, I do not know, but the local segregation resulting in hard spots near the outer surface is undoubtedly due to bad practice somewhere before the steel reaches the mold. The condition before casting must be right within narrow limits, as regards temperature and degree of oxidation as well as composition, otherwise the results cannot be good.

A blow-hole, in a way, is the result of local segregation of gases, and is often, if not always, accompanied by segregation of other impurities as well, particularly carbon. In forgings made direct from the ingot and with insufficient work, and having a large number of hard spots, I have seen close beside each hard spot the remains of a blow-hole which had not been wholly closed up by the forging-process.

Regarding the two supposititious methods of solidification of steel ingots described by Professor Howe, the one assuming the formation of "pine-tree" crystals is attractive, since if they are large it gives a reasonable explanation of the local segregation which results in numerous hard spots near the outer skin of the ingot by entangling or inclosing portions of the liquid steel, which is in condition for segregating as it freezes. Such crystals may occur during solidification in certain grades of steel made in certain ways (probably not the best), but they cannot be assumed to be formed always, especially in the soft grades. It seems probable that they will be found, if at all, only in the harder grades and in those that are not well made.

Granting that continuity of structure is essential in the best steel ingots, the ideal procedure to make them is to start with suitable plant and materials, make the steel properly as to physical and chemical conditions before casting, and reduce or obliterate the pipe and the central segregation by compressing the ingot laterally while the interior is still fluid. It is assumed, of course, that all details favoring these broad divisions of the operations will be adopted. The possibilities of lateral compression have not as yet been fully realized.

To speak of the entrained silica, silicates, and oxides in steel as slag is, in a sense, misleading, as while they may be the same as some of the slag-ingredients in composition, and should properly find their way into the slag, they are not wholly derived from the slag. To call them slag is much like calling springs along the banks, the water from which flows into a river, the river itself.

To remove the entrained oxidation-products from steel, agitation is more efficient than time. The individual particles of oxides and silicates are so minute that gravity alone does not free the steel of them much faster than it clears rily or oily water. The operation of removing them is thought to be like this: Assuming that they are fusible at the temperature of the molten steel, each particle will, when it touches another, stick to it, forming a larger particle, which again will unite with another, and this operation continues until the resulting particle becomes large enough to float to the top and merge into the slag. This will be very slow if effected by gravity alone. By stirring, the time required for the elimination of these products to any given degree may be reduced to a fraction of that required by the action of gravity alone. The agitation which takes place in the molds in soft steel made by the evolution-method, due to the escaping gases, is very efficacious to the end in question.

If, however, the oxides or silicates are infusible at the temperature of the steel, so that they do not "wet" each other and unite when they touch, there will be imperfect elimination of them, limited almost wholly to that effected by the cleansing absorption by the molten slag of such particles as come in contact with it, and the steel will be red-short. Silica alone, or magnetic iron oxide alone, undoubtedly acts in the manner described, and probably largely so when they exist together in steel. Manganese silicate, however, is the great agent for the removal of the entrained oxidation-products by fluxing them, and this is formed from the final additions and entrained silica.

In the open-hearth furnace the charge may be retained until it shows the proper qualities for casting. The oxidizing conditions may be reduced by appropriate treatment to almost any degree, so that the metal approximates very closely to

crucible- and electric-furnace steel. The temperature also may be brought to the desired degree by raising or lowering it as needed.

This, more than anything else, except perhaps the increased yield, will bring to pass the fulfillment of the often-quoted prophecy about the open-hearth process attending the funeral of the Bessemer process, and gives the former a great advantage over the latter process for making certain grades of steel. In the Bessemer process both the degree of oxidation and the casting-temperature must "be caught on the fly," as it were, and after the converter is turned down but little can be done to change either of these fundamental conditions.

Henry M. Howe, New York, N. Y. (communication to the Secretary*):—Dr. Dudley, in speaking of the genesis of the pipe (p. 821), has "never found an instance in which the metal had separated after being firm." But how can he know this? On pp. 36 and 48 of my paper I gave my reasons for inferring from the shape and structure of certain parts of certain pipes, that those parts had opened in metal already firm. Does Dr. Dudley not find that shape and structure, or does he not agree with my reasoning?

Mr. Stevenson seems to imagine several differences of opinion between himself and me which either do not exist or are immaterial.

As to the shape of the molten lake during the solidification of the ingot, the difference between Mr. Stevenson's idea and mine seems to me wholly immaterial. The bottom of the lake will of course be nearly flat at first, as he sketches it. The pointing which I sketched would come much later. At what particular moment it comes is not material to anything in my discussion, so far as I see; nor is it material that it should come at any time.

Nor have I any objection to supposing that imprisoned gases may aid in the support of the bridges. This idea does not conflict with my own in the least. A bridge cannot be supported by gases until it is a solid bridge. When solid its own rigidity is surely competent to give it some support. If gas aids this rigidity, so much the better.

* Received May 5, 1908.

Mr. Stevenson's solidification (p. 834) is "invariably" of the land-locking type. How does he know?

Mr. Stevenson agrees with me as to the effect of (1) casting with the large end up, (2) top-casting, (3) sink-head, (4) slow pouring, and (5) aluminum. Dr. Dudley agrees with me in the first and last of these, and does not disagree, so far as I see, in any respect. That he agrees as to slow pouring is to be inferred from his saying that the pipe is shortened by the use of a small nozzle. Mr. Stevenson differs with me as regards the effect of the width of ingot and the rate of cooling (use of sand *vs.* iron molds) on the depth of the pipe. There are so many variables that unless great pains are taken the effect of the one we have in mind may easily be masked by that of others. The use of a baked sand mold instead of an iron one may so far restrain the evolution of gas and thereby the formation of blow-holes, and thereby increase the depth of the pipe, as in this indirect way to outweigh its direct effect of shortening the pipe by increasing sagging. In one direct series of experiment of which I know, the depth of the pipe actually decreased with the width of the ingot, as my prediction calls for. Beyond this, in a great number of very narrow ingots within my own knowledge, the pipe has reached very much deeper than it would have gone had the ingots been wider. But there may be other influences in addition to those which I have considered which under certain conditions may outweigh mine.

It is not with regard to piping and segregation, but with regard to the use of the English language, that Mr. Stevenson differs with me as to whether the casting-temperature should lie "as low as practicable," or "as low as advisable." I see no serious objection to substituting "advisable" for "practicable," though to me "practicable" seems to convey more clearly the idea of being as low as is permitted by the attendant conditions, such as the need of running freely through the runners, not leaving a heavy ladle-skull, etc. But is this sort of criticism quite worth while?

I take issue very squarely with Mr. Stevenson as regards the supposed conflict between low casting-temperature and slow pouring. I see no conflict. To pour slowly and yet cool, merely means to pour into many molds at the same time, ex-

actly what Mr. Stevenson himself is doing, for this among other purposes, in bottom or rather group-pouring.

As regards the size of ingots, Mr. Stevenson and I agree, as my paper presented in the morning before his remarks clearly shows. If anybody proved by abundant direct experiments, before the publication of my paper,[2] that ingot-size and rate of cooling may have but slight effect on the degree of segregation, I do not know it.

Mr. Stevenson seems to agree with me that deep-seated blow-holes will lessen the pipe. I certainly agree with him that they should not be tolerated in high-carbon tire-steel, because they would not weld, and, unwelded, would be disastrous by causing shelliness. That remedy is of value in many, but certainly not in all cases.

In short, though Mr. Stevenson might lead some to suppose that we differ in nearly every respect, in point of fact our differences are very slight.

Professor Campbell is right in what he says about finishing-temperature. It is only on the assumption that by "finishing-temperature" we mean that at which distortion ends or ceases to be severe enough to unequiaxe the existing grains so far as to cause a new grain to form corresponding in size to the then-existing temperature, that the laws defining the relation between finishing-temperature and quality can be supposed to be true. This all of us should have had, and many of us have had, in mind; but Professor Campbell does well to call our attention clearly to the matter.

Genesis of the Lake Valley, New Mexico, Silver-Deposits.

Discussion of the paper of Charles R. Keyes, p. 139.

Bernard MacDonald, Guanajuato, Mexico (communication to the Secretary*):—Mr. Keyes's paper is very interesting to me because of my personal experience with the development of the ore-deposits of Lake Valley.

[2] A Further Study of Segregation in Ingots, *Engineering and Mining Journal*, vol. lxxxiv., No. 22, p. 1011 (Nov. 30, 1907).

* Received July 30, 1908.

Seeing that many of the important facts connected with the early history of the development of these mines are either now unknown, or have become more or less distorted, and thinking that it would be interesting as well as valuable to have these recorded as a historical supplement to Mr. Keyes's elaborate paper, I present the following narrative:

In February, 1881, soon after the Atchison, Topeka & Santa Fé railroad made connection with the Southern Pacific at Deming, N. M., the Lake Valley properties were brought to the attention of New York and Philadelphia capitalists by John A. Miller, of Silver City, N. M. At that time silver was $1.10 an ounce and silver-mining was very attractive. Vast fortunes were being made in Leadville and other Colorado camps, and Tombstone was pouring out its millions. Lake Valley, like all other mining-camps in their early history, had its drawbacks, but the railroad, then passing within 14 miles of it, mitigated many of these. The camp lay in the foot-hills of the Black range, in which the Apaches had their stronghold, and raided, when it suited their convenience, the surrounding country, killing the prospectors and pioneer settlers and stealing their cattle. This band of Indians was joined by the outlaws and desperadoes from the frontiers of Texas, Arizona, and Mexico, who collectively defied the United States soldiers, restrained, as the latter were, by the mollycoddles from the East. The brutality of the Indian raids through Arizona and New Mexico in those days would scarcely be believed now, but they stain in deep scarlet the early records of these territories, and are still well remembered by the early residents. These conditions, which prevailed in southwestern New Mexico and eastern Arizona in 1881, constituted the principal drawback to mining operations in the frontier camps. The prospective purchasers hesitated before these problems: are the mines valuable; and if purchased, can they be worked profitably under existing conditions? But the representations of the properties were attractive, and the mining-fever was abroad in the land. The services of George Daly, a mining engineer and operator of national reputation, then manager of the famous Robinson mine, at Kokomo, Colo., were secured for the purpose of reporting on the merits of the mines and the surrounding conditions.

After concluding his examination, Mr. Daly went to New

York and made his report in person, and, as this report was favorable, arrangements were made to purchase the properties. The price, including commissions, was said to be $300,000, of which John A. Miller received $100,000, McEverts, Lufkin, and the other owners the balance. George W. Lufkin had only a small interest, for which he received $15,000. Mr. Lufkin was a prospector and a fearless Indian-fighter, but I do not think he was a mining engineer, as has been stated. I knew him personally, and saw him occasionally for two years afterwards. His cabin, consisting of one room, was the only house at the mines at the time of the purchase.

The New York people interested in the purchase were represented by George D. Roberts, a mining-man of California and Nevada, and the Philadelphia parties were represented by Whitaker Wright, who was prominent as a mine-operator in Leadville at that time, and whose trial and suicide in London recently attracted so much attention.

The claims included in the purchase covered about 5,000 ft. in length, and comprised within the boundaries all the "outcrop" visible in the district, and, by the way, all the ore-deposits that were ever found.

After the purchase was arranged, Daly returned to Leadville to arrange his affairs, so as to give his personal attention to the operation of the properties. At this time I was superintendent of the Iowa Gulch Mining Co., and Daly offered me a similar position at the Lake Valley properties, which, owing to special inducements, coupled with a glowing description of the rich, undeveloped new region, I was glad to accept. I left Leadville April 20, 1881, and arrived at Lake Valley three days later.

I went over the properties with Mr. Daly, who explained his theory of the genesis and localization of the ore-bodies, from which we developed a working hypothesis for their development. The theory (a modification of the one which accounted for the Leadville ore-deposits) was that the ore lay in contact between two dissimilar lime strata, a fossiliferous fissile white lime and a heavy-bedded dark-grayish blue lime, which were supposed to constitute the hanging- and foot-walls respectively. It was assumed that the contact-plane between these strata was opened up during the eruption of the "porphyries" which

constituted the high hills to the west, and gave an opportunity for the flow of the mineral solutions and the deposition of the ore-bodies in contact between these strata.

Unfortunately for the permanence of the ore-bodies, this theory was proved to be incorrect by the early development-work, as the ore-bodies were found to lie in channels of very limited depth in the "blue lime," and their source was not from below.

At that time the deepest development on the ore was an inclined shaft sunk on an ore-channel on the Columbia claim (subsequently included with the properties taken over by the Sierra Bella Co.) to the depth of 75 ft. This shaft, with an open-cut on the "Grande" 25 ft. long, reaching a depth of 12 ft., and numerous shallow cuts, none of which exceeded 5 or 6 ft. deep, was the extent of the development. Most of these openings, however, showed "pay" ore in greater or less quantity, which, in connection with an outcrop, traceable at intervals for a length of 3,000 ft. along the line of contact above mentioned, constituted a good prospect.

A few days later Daly returned to New York to assist in the organization of four companies to take over and operate the properties—namely, the Sierra Apache Co., taking the easterly claims of the group; the Sierra Bella Co., taking the claims next adjoining to the west; the Sierra Grande Co., taking in the claims adjoining further to the west, and the Sierra (name not remembered) Co., taking in the claims on the extreme west end of the group. Daly owned a fourth interest in these companies, the remaining interests being owned by the New York and Philadelphia people.

Development was begun on the Sierra Grande and Sierra Bella properties (these showing the more favorable indications), in accordance with the theory above stated, by commencing a row of shafts at a distance of about 75 ft. from the outcrop in the direction of the dip. However, when one of these shafts passed through the fossiliferous lime, and entered the "blue" lime, supposed to constitute the hanging- and foot-walls respectively of the ore, without finding any ore, or even the gouge-matter of a movement-slip, the contact-theory, so far as it implied the indefinite downward and horizontal continuation of the ore between these two lime-deposits, was abandoned, and

development was then carried on under the old unscientific but much to be recommended method (especially in new districts where the habit of the ore-deposit is not yet known) of "following the ore."

The results of this method were all that could be expected, for in a few months the good and the bad points of the ore-deposits were determined. These deposits were found to lie in troughs or channels in the "blue" lime, which seemed in most cases to have been caverned out for their reception, and were capped over, except very near their outcrop, by the "fossiliferous" lime, which, as I recollect, had an average dip of about 10° to the south, and which formed the upward boundary of the ore except in a few cases, as that in the Columbia claim, where the ore-deposit left the contact above referred to, and dipped down about 45° in caverns in the "blue" lime.

The only departure we made from the rule of following the ore was the sinking of a "joint" shaft, which reached the ore at the depth of 25 ft., and in cutting the ore-deposit at this place the phenomenal deposit of cerargyrite (horn silver), which we called the "Bridal Chamber," was discovered. The history of this development, which afterwards became famous, was as follows:

A drift, called the outcrop drift, was first started as an open-cut in the ore, and was being driven westerly parallel with the outcrop, but from 25 to 35 ft. below it; at this time the drift had attained the length of 800 ft., all in ore, and had shown the existence of a large deposit of ore, the value of which kept increasing as the drift was extended. In order to hasten the development a shaft was sunk about 100 ft. in advance of the drift, so as to determine if the ore extended that distance. In addition, this shaft was to be used to ventilate the workings and facilitate the further development of the ore.

The site selected for this shaft was on the dividing-line between the claims owned by the Sierra Grande Co. and the company adjoining it to the west, and for this reason it was called the "joint" shaft, both companies being charged with its cost. The shaft site and for a considerable distance around it was covered with a thicket of catclaw brush, which was cleared away and the shaft commenced early in August.

The shaft passed through about 4 ft. of detritus and 20 ft. of

the hanging-wall lime, a total of 24 ft. of overlying material, before entering the ore-deposit, which was 25 ft. thick at that place. The vertical section of this shaft showed:

	Feet.
Detritus,	4
Fossiliferous lime stained with the oxides of iron and manganese,	20
Ore averaging 40 oz.,	5
Ore averaging 60 oz.,	3
Ore averaging 150 oz.,	4
Open space,	1½
Horn silver, averaging 15,900 oz.,	4
Lead (sand) carbonates averaging 500 oz. silver and 40 per cent. of lead,	5
Decomposed lime heavily stained with iron and manganese,	3
Total thickness of deposit,	25
"Blue" lime,	below (undetermined)

The extraordinary commercial value of the deposit developed at this place was due to the occurrence of the streak, 4 ft. thick, of nearly pure cerargyrite. The commercial value and physical features of this ore-body made it and the Lake Valley mining region famous. The "outcrop" drift above referred to connected with this shaft showed a continuous trough of ore 900 ft. long.

The downward continuation of this ore as a continuous sheet or bed was of limited extent, except for the narrow pipes, which extended for considerable distances in the "blue" lime, but these finally feathered out and ceased entirely without leaving even a seam, showing that the dissolution of the "blue" lime and its replacement by the ore were simultaneous.

An unfortunate circumstance in connection with these mines and the discovery of the bonanza ore-deposits in them was the strange coincidence, that on the very day, and almost the very hour, at which the "joint" shaft broke the 4-ft. streak of horn silver, George Daly was killed by the Indians. He was pursuing, with a posse of citizens and miners, a band which had raided the town of Hillsboro and killed several men and women.

I named the ore-deposit cut in the "joint" shaft the "Bridal Chamber" because of the sparkling light reflected by the myriads of crystals of cerargyrite and calcite studding the roof of the open space over the chloride streak. The purest specimens of the chloride streak assayed 20,000 oz. of silver to the ton, and the average across the 4-ft. face was 15,000 oz.

If the flame of the candle was held against any portion of this streak the ore would decrepitate in the flame and fuse to a slag. The streak of sand carbonates that underlaid the chloride deposit was practically co-terminous with it, and as soon as the chloride streak broke up and disappeared the lead carbonates disappeared also. At a radius of 15 or 20 feet from the point cut by the "joint" shaft, the chloride streak broke up into small streaks and segregations, but the ore continued to average from 200 to 500 oz. for a considerable distance beyond this point, gradually lowering still further as the edge of the deposit was approached.

Belá Kadish was the first assayer on the properties. I retired from the management at the end of the year 1881, D. R. Jackson then becoming its manager. He built the 20-stamp silver-amalgamation mill for the treatment of the ore, which operated for a time on the rich ore, but was not much of a metallurgical success. Jackson was followed by Dr. Endlich, who shipped all the ore extracted from the mines, including all the mill-tailings, to the smelter at Pueblo, Colo. The tailings averaged about 250 oz. to the ton. Endlich was followed by Walter C. Hadley, as manager, during whose administration the ore-bodies were practically exhausted, the total yield, I believe, being approximately 7,000,000 oz. of silver.

The foregoing is my recollection of some of the prominent events of the sad, strange, and unique history of the mining-camp of Lake Valley, and one of my reasons for recounting this narrative is that Mr. Keyes has made no mention of Mr. Daly's connection with the properties.

Present Mining Conditions on the Rand.

Discussion of the paper of Thomas H. Leggett, p. 211.

Alfred James,* London, Eng.:—I thank you, gentlemen, for your kind invitation to address you. It is a very great pleasure for me to be here at your annual meeting, and, although I have been a member since 1894, this is the first opportunity I have

* President of the Institution of Mining and Metallurgy.

had to attend a meeting of the Institute. Perhaps it is all the more a fortunate coincidence that I am here now, because it so happens this year that you have as your President a very distinguished mining engineer, who is representative of the body of engineers who have done so much for the Rand; and, besides, you have been fortunate in having presented to you a paper from Mr. Leggett, one of those very engineers to whom South Africa is so much indebted.

Twenty years ago mining was not the sound commercial undertaking that it has since become; but now, with high-grade engineers leaving as little as possible to chance, the stage to which Mr. Leggett refers has been attained. And here I want you to notice particularly that the Rand is now producing 25 per cent. more than the world's total output of gold when the American engineer first went there. The value of the total output of gold in the world during 1888 was $110,196,915, while the value of the output of the Rand alone for 1907 was about $128,750,000, which shows the progress that has been made.

With regard to some of the points to which Mr. Leggett has specifically referred, I was in South Africa in January of this year, and I was surprised to find the cheerful feeling concerning labor, in spite of having to return the Chinese laborer home. This optimism was remarkable, because a native is engaged for six months only, and then he goes away, while a Chinaman is held for three years and trained to become a highly proficient worker. Contrary to the general expectation, however, it was found at the end of December that quite a large body of natives was ready and willing to work, and, as Mr. Leggett pointed out, it was a strange sight to witness these natives begging for work instead of having them brought in under contract by contractors, as was formerly the case. There were, of course, various reasons for this new condition; one is, that during the war there was a big demand for natives and they got very well paid, but they have been living on that money since the war and have spent it, and now they are seeking work. Immediately after the war there was a dearth of native labor, but now the natives are coming back to earn more money.

With regard to the future production of the Rand, it may

not continue to increase as rapidly as it has in the past. If you will review the technical press of a few years back you will find that it was seriously stated that it paid best to work out everything in the quickest time possible, because of the saving of interest on capital from the more rapid distribution of profits, as well as from lessened costs arising from working on a larger scale. This reasoning, if carried out to its logical conclusion, would show that it would pay best to work out the whole earth in ten years. I think the Boer government is not going to favor this view. The order of the day is to husband all the money that has been subscribed, and to have it used most profitably. I think the government will desire to keep the mines going at a reasonable and decent rate for many years to come, so that, in the meantime, the other industries of the country will have a chance to develop.

I was surprised on my recent visit there to find that so many men were running farms successfully in the Cape Town districts. There has been a collapse in trade, but the agricultural districts are doing well. If, by means of the gold-industry, aid is given to the agricultural and other interests of the Colony, it is really good business, from a political point of view, not to have the gold output too rapidly developed.

W. Fischer Wilkinson, London, Eng. (communication to the Secretary*): In this paper Mr. Leggett discusses the capitalization of the mines, and expresses the opinion that it is demonstrable that the majority of the companies are not overcapitalized. If his calculations are founded on the relation between the earnings and the nominal capital, as might be inferred from the examples given, the argument is unsound. Companies like the Crown Reef and the Ferreira, which belong to the original outcrop mines, are, of course, paying a handsome return on their original capital upon whatever basis it may be calculated. But in the case of most companies formed since 1895, if regard be paid to real rather than to nominal capital, Mr. Leggett's thesis becomes doubtful. Those, for instance, who subscribed in 1902 for Knights Central shares at £3 or for South Knights at 35s.—to mention two companies, referred to in the paper, which are now obliged to amalgamate, owing

* Received June 22, 1908.

to insufficient working-capital having been in both cases originally provided—might reasonably claim, I think, that these mines were over-capitalized.

The true capital, upon which interest must be earned, is fixed by the price at which the working-capital shares were issued to the public. The nominal capital is a fancy figure, which in most cases is far below its real value.

On studying any directory of the Transvaal mines, it will be found that many of the mines have issued working-capital shares at a high premium, and big dividends on the nominal capital have to be reduced proportionately.

To illustrate the sort of company formed, I will assume that the ground was considered to be worth £500,000 and that £500,000 was required for working-capital. Such a company would not have a capital of 1,000,000 shares of £1 face-value, but perhaps of 200,000 £1 shares, of which half would be taken by the vendors, while the other half would go to the subscribers of cash at £5 per share. The company would, of course, have to earn dividends on a capital of £1,000,000, and not on the nominal capital of £200,000.

Whether or not the majority of the mines would show a fair return on the capital invested I do not know. I only wish to point out that it is the cash subscribed, and not the nominal capital, that should be the basis of the calculation.

The Work of the Testing Department of the Watertown Arsenal, in Its Relation to the Metallurgy of Steel.

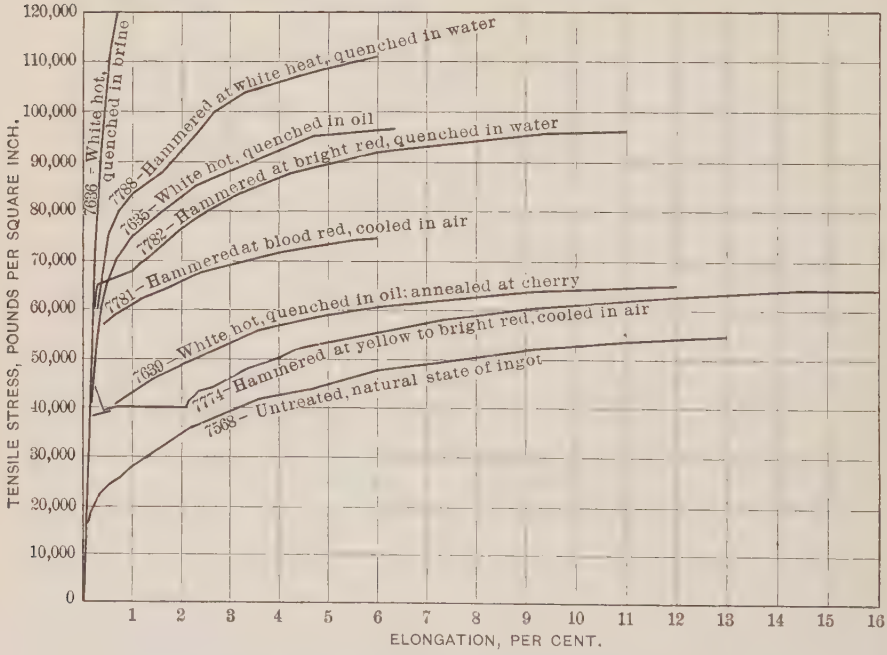
Discussion of the paper of James E. Howard, p. 223.

James E. Howard, Watertown, Mass.:—In connection with this paper, it is the desire of the Watertown laboratory to receive suggestions as to the lines of work and the particular direction along which the examination of the ingot-metal should proceed, and any suggestions so offered will be noted with care, and the tests conducted so as to embrace such features as it is thought desirable to have investigated.

To me the question of structural continuity seems of para-

mount importance in those investigations of the properties of steel which begin at the ingot. The modification of the tensile properties may easily be accomplished in the mechanical treatment which succeeds the ingot state of the metal, as is illustrated in Fig. 1, showing the tensile properties found in low-carbon steel.

It will afford us satisfaction to have some specific or general procedure for conducting the test indorsed by the members of the Institute. The work will necessarily take a definite direc-


TENSILE STRESS, POUNDS PER SQUARE INCH.
120,000
110,000
100,000
90,000
80,000
70,000
60,000
50,000
40,000
30,000
20,000
10,000
0
1 2 3 4 5 6 7 8 9 10 11 12 13 14 15 16
ELONGATION, PER CENT.
7636 - White hot, quenched in brine
7788 - Hammered at white heat, quenched in water
7635 - White hot, quenched in oil
7782 - Hammered at bright red, quenched in water
7781 - Hammered at blood red, cooled in air
7630 - White hot, quenched in oil; annealed at cherry
7774 - Hammered at yellow to bright red, cooled in air
7568 - Untreated, natural state of ingot

Chemical Composition: Carbon, 0.20; Manganese, 0.58; Silicon, 0.015; Phosphorus, 0.017.

FIG. 1.—RANGE IN TENSILE PROPERTIES OF STEEL FROM THE SAME INGOT, DUE TO HEAT AND MECHANICAL TREATMENT.

tion, and a number of different lines have already been suggested. The question of the manufacture of steel rails, which has attracted as much attention as any other subject, is now being studied at the Watertown Arsenal.

R. W. MAHON, West Albany, N. Y. (communication to the Secretary*):—An investigation of ingot-structure and the

* Received Feb. 12, 1908.

causes leading thereto, and of the effects in the rail or other finished shape following the various observed features of ingot-structure, would be in effect a study of steel metallurgy. If such an investigation is to be carried out in an exhaustive manner, I think that, before proceeding to any study of finished material, the investigation should begin with a study of the ingot. Besides this possibly more scientific and fundamental way of proceeding, I wish to suggest that there are many problems the investigation of which could be carried out, principally by a study of finished material. This investigation might proceed simultaneously with the proposed more elaborate scheme, and thus shorten the road to the practical results desired by the users of steel. I propose the following subjects, which could well form a part of the general program:

1. *Tires.*—The cause of the shelling of the tread. What precautions during the process of manufacture will prevent shelling in service? What is the best composition to prevent shelling?

A peculiar thing I have noticed for years is that the trailing tires and the tires used on tenders are the ones which develop those shelling-defects the most frequently. Furthermore, it is new tires which fail from shelling, not those which have been turned after service. This fact suggests the idea that shelling may have its primary cause in unusually columnar or unsound exterior ingot-surface: this being an application of Dr. P. H. Dudley's idea, that rail-heads sometimes fail from this cause. I do not think that the cause of the failure of the trailing-tires is the great weight on them, because the weight is no greater than it is on the driving-wheel tires.

2. *Steel Castings.*—The best results in annealing steel castings are obtained when they are heated to between 700° and 800° or possibly 850° C. What is the best temperature at which to remove steel castings from the furnace in which they are being annealed?

Is the range from 700° to 850° C. the best annealing-temperature for vanadium-steel castings?

What is the best temperature at which to remove vanadium-steel castings from the furnace in which they are being annealed?

A manufacturer of steel castings in a large way recently told

me of certain small frames that he was trying to make. He could not make them according to the specifications, and finally the superintendent went to the office of the works to say that he would have to give it up. There was a heat in the furnace, and during his absence the foreman, in exasperation, took hold of the castings and threw them on the floor. A few days afterwards the steel was noticed to be very elastic. Test-pieces were prepared from it, and the material was found to meet the specifications perfectly. The superintendent corrected his first statement and said that he could make them. Now that was only a question as to the temperature at which that particular steel casting should be removed from the furnace in order to anneal it properly.

3. *Steel Plate.*—Some people are skeptical of obtaining the same degree of homogeneity in plates rolled from small ingots and in those rolled from slabs after the proper discard has been made. As this is a matter of fundamental importance to all steam-users, I propose an additional subject for investigation, as follows: a thorough investigation of the question of the homogeneity of plates rolled from the ingot and plates rolled from the slab.

4. *Axles.*—One of the most vitally important materials purchased by the railroads is axles. While in making other steel products for the roads there is some degree of uniformity in methods of procedure among manufacturers, in making axles the contrary is true. Axles are partly shaped in rolls and then rapidly forged at one heat, not being subsequently annealed. They are made entirely by rolling without subsequent forging. They are forged at two heats, by alternately heating the two ends, in each case heating more than one-half of the length of the axle, and not subsequently annealed. They are forged and then annealed by the Coffin process. They are forged and annealed in an annealing-furnace, being held in the furnace until cold. Sometimes axles after this last operation are of unsatisfactory character, due to lack of care or skill, and the annealing is repeated. So that a lack of uniformity in the process may truly be said to exist. In consequence of these facts I propose that two additional subjects be added to the investigation. Of course, this would not be altogether an investigation into the unknown, but in some directions a striking demonstration of well-known facts.

1. What is the best method of shaping and annealing steel axles; and what is the effect upon the value of the finished product of each of the modifications at present in considerable use in this country? and

2. What is the effect of repeated annealing?

A. A. Stevenson, Burnham, Pa.:—There is one very important point that I have not heard mentioned by Mr. Howard, and that is the question of the minimum amount of work for maximum effect. Is this not a question well worth investigation? Is it not possible that there are cases where there is too much work put upon steel, and could not as good or better results be obtained by a smaller amount of work?

Now as to the question as to what led up to the meeting which was held in New York, Sept. 24, 1907: An appropriation was made by the government some time ago for use in experimental work at Watertown Arsenal. William R. Webster and Professor Marburg were appointed by the government to act in connection with the Watertown Arsenal authorities in an advisory capacity.

Unless I am mistaken, the meeting was largely the result of some missionary work on the part of Mr. Webster and Professor Marburg. Letters were written to various engineers, railroad men, and manufacturers. The meeting itself was purely informal. The object of the meeting was to talk over the work that might be done with the appropriation, and Major Ruggles and Mr. Howard were anxious to obtain suggestions as to the lines of work which might be followed that would be of greatest use to the engineering profession, and to ascertain whether contributions of material for experimental purposes would be made.

A record was kept of the proceedings, but the idea at that time was not to have this record published. The meeting itself was very interesting. There was a free expression of opinion, and numerous suggestions were made. There seemed to be a little feeling on the part of some of the speakers that the work to be done was in a sense superfluous. The question was asked, "Does this mean that we will shut down our own laboratories?" The majority felt that this was not the idea at all, and that the work at Watertown Arsenal would not take

the place of the work in our own laboratories; the experiments to be conducted there would not take the place of those conducted in our own shops, but rather would supplement them.

I think manufacturers should assist in every way the work that has been started by this appropriation. Several steel-makers have already made contributions of material to Watertown Arsenal and work is now in progress. None of the results are yet ready for publication.

J. A. KINKEAD, Schenectady, N. Y.:—I made a comparative test of longitudinal and transverse sections of steel forgings recently, with the result that the transverse test gave 13 per cent. elongation and 11 per cent. reduction, while the longitudinal test gave 27 per cent. elongation and 30 per cent. reduction.

In regard to steel castings, a few years ago the Bethlehem Steel Co. reported some very remarkable tests of nickel-steel; and one of our steel-casting friends took his ordinary product and by proper heat-treatment duplicated the results in every respect; in other words, where they had a 0.40-carbon nickel-steel he used 0.40-carbon cast-steel, and by proper annealing the same elongation and reduction of area were obtained.

In regard to transverse tests that have been made on pieces of rails, we meet that all the time in boiler- and fire-box plate, and in many cases the transverse tests equal the longitudinal. In some instances we have cut four sets of tests, and the tranverse tests have averaged as good as the longitudinal. Of course, the longitudinal tests were affected to some extent by the rolling of the shell, which would affect the transverse test to a less extent. It is understood that all fire-box steel is cross-rolled.

I have recently had a case of defective steel in which the tensile tests were satisfactory in every way, yet the castings had more or less cracks and blow-holes that could not be accounted for by the steel-foundry. On investigation it was shown that the charge for the open-hearth furnace was not what it should have been.

On the subject of billets, I could get any quantity of exhibits of that kind. In using up stock billets it is necessary to cut many to the right length in cold saws, which readily shows piped centers.

CHARLES L. HUSTON, Coatesville, Pa.:—Mr. Howard's point is well taken, in his statement that a general knowledge of the condition of the metal in the ingot is highly essential to the study of the material and its fitness for use after having been worked into shape.

Variation in yield-point, elongation and reduction of area, however, may be produced by the manner of heating and manipulation in the rolls, as also by the proportioning of the test-pieces.

I suggest that for the study of the ingot-structure, in addition to the proposed division by grades, such as open-hearth, Bessemer, and crucible steels, they should be carefully divided into mild steels, steels of medium carbon, and steels of high carbon, and also the different alloy-steels, on account of the very different action of these different steels in the process of cooling and solidification from the molten state, which action largely affects the question of chemical uniformity as well as of structural continuity.

In the proposed observance of working-temperature, number of pieces, etc., I believe that little effect will be observed in the final product from work done at the higher temperatures above the point of recalescence, as the work done at these temperatures does not permanently affect the structure of the steel to any great degree.

I apprehend it will be found that the best and most practical results will be gained by ordinary "horse-sense" methods, carried out in patient, painstaking study of all stages of the processes of manufacture and manipulation and their effect on the final product.

I am glad, indeed, that this subject is now being taken up vigorously by investigators of steel; and while it means a lot of trouble and probably will result through time in disagreement between the different "Doctors," yet I think ultimately it must result in a better appreciation of the conditions we have to deal with in steel, and in obtaining a more practical and satisfactory basis of specifications and inspection.

I want to encourage the use of a softer range of steel for boiler and structural work, as I believe that many of the specifications now in use run to the high limit too much and endanger the use of brittle steel, especially as in plates the

portions usually tested are not the hardest portion of the steel in the plate, and I believe that where softer steels are used a closer factor of safety would be warranted.

I believe that, in practice, better results would be derived if, instead of allowing pressures or stresses in proportion to the actual minimum tensile strength obtained, which introduces a temptation to use just as hard steel as can be passed through under the inspection and thus avoid the use of greater thickness in order to carry heavier stresses, the thing were gotten at in the reverse way by establishing the unit stresses in the finished structure and then specifying the range of tensile strength of steel which will be allowed to stand these unit stresses; for instance, given a unit stress of 12,000 lb. per sq. in., permit the use of steel anywhere between 50,000 and 60,000 lb. per sq. in., provided it has a ductility of 1,500,000 divided by the tensile strength in pounds, thus using the combination of tensile strength and ductility to determine the maximum unit stress, instead of the tensile strength alone. This would encourage the use of softer steels for the same work, and I believe would result in much greater safety to the traveling public, who intrust themselves to the skill and fidelity of the engineers and manufacturers responsible for the structure which carries them.

F. N. Speller, Pittsburg, Pa.:—Regarding the matter dealt with in Mr. Howard's paper, I am inclined to agree with Mr. Stevenson that all inquiries into the origin of defects in steel must go back further than the ingot, starting with the metal from the blast-furnace and giving due regard to its composition, treatment in refining, ladle-reactions, deoxidization, etc. For example, in the manufacture of soft welding steel we may add sufficient ferromanganese to deoxidize the metal, and, by a good reaction in the ladle, flush out the steel so that the oxides formed are practically all absorbed by the slag. A slight change in the relative proportions of silicon, manganese or other elements in the pig-iron may alter conditions entirely, giving a ladle-cinder incapable of absorbing the products of deoxidation from the steel, and resulting in more or less trouble all the way to the finished product. Yet, in this case, the chemical composition and structure of the ingot may be

normal; the steel, however, is liable to develop defects under subsequent mechanical treatment.

The fact that lines of weakness are developed under stresses applied transversely to the direction of rolling does not necessarily prove the metal to have been structurally unsound in the ingot; to my mind, it is more often evidence of lack of lateral work. We have made a close study of soft Bessemer steel used in the manufacture of welded pipe, having, I presume, one of the best opportunities for studying this class of steel, since our practice lies in the manufacture of this steel exclusively. It has been our experience that most, if not all, structural defects in ingots are overcome by proper treatment in rolling; however, we are open to any advantages which the carefully-planned tests outlined in Mr. Howard's paper offer in the way of learning more about steel in general, and to this end will be glad to co-operate with the Watertown Arsenal Committee so far as possible.

E. H. McHenry, New Haven, Conn. (communication to the Secretary*):—I have taken great interest in this matter at all times, but in reading recent discussions on the subject, I find that the suggestions and recommendations which I might wish to present have been anticipated almost completely, and I do not feel that I can add information of value at present.

I wish to suggest, however, that photomicrographs from the crystalline structure be added to rail-specifications, preferably in double form, showing average requirements and the extreme limits in the direction of coarse crystallization. Four or five years ago I secured the introduction of such matter in the specifications of the Canadian Pacific railway, in a contract awarded to the Pennsylvania Steel Co. for high-carbon rails, rolled at the Sparrow's Point works.

I also suggest the determination of the value of carbon in terms of structural stiffness, considering the rail as a girder, as I have long believed that a relatively light high-carbon rail, with the same deflection as a heavier section low-carbon rail, is a more economical method of securing the requisite strength. The results could be tabulated, showing equivalents in rail-sections and carbon-percentage at intersection of columns.

* Received Mar. 31, 1908, through the courtesy of William R. Webster, Philadelphia, Pa.

The same considerations apply in much greater measure to angle-bars, as conditions do not readily permit the addition of metal at any point excepting the neutral axis, where it is least effective.

I also suggest the addition of an abrasion-test to determine the loss of weight and penetration, which may possibly be best attained by noting the effect of revolutions of a hardened steel bar applied to the rail under high pressures.

I believe these last two points are particularly worthy of consideration, as the two principal and almost the only functions of the rail are those of a girder and of a wearing-surface. To the best of my knowledge, modern specifications make no provision for determining either of these important points, apart from the somewhat doubtful results obtained from the drop-test, which is primarily for the purpose of determining brittleness.

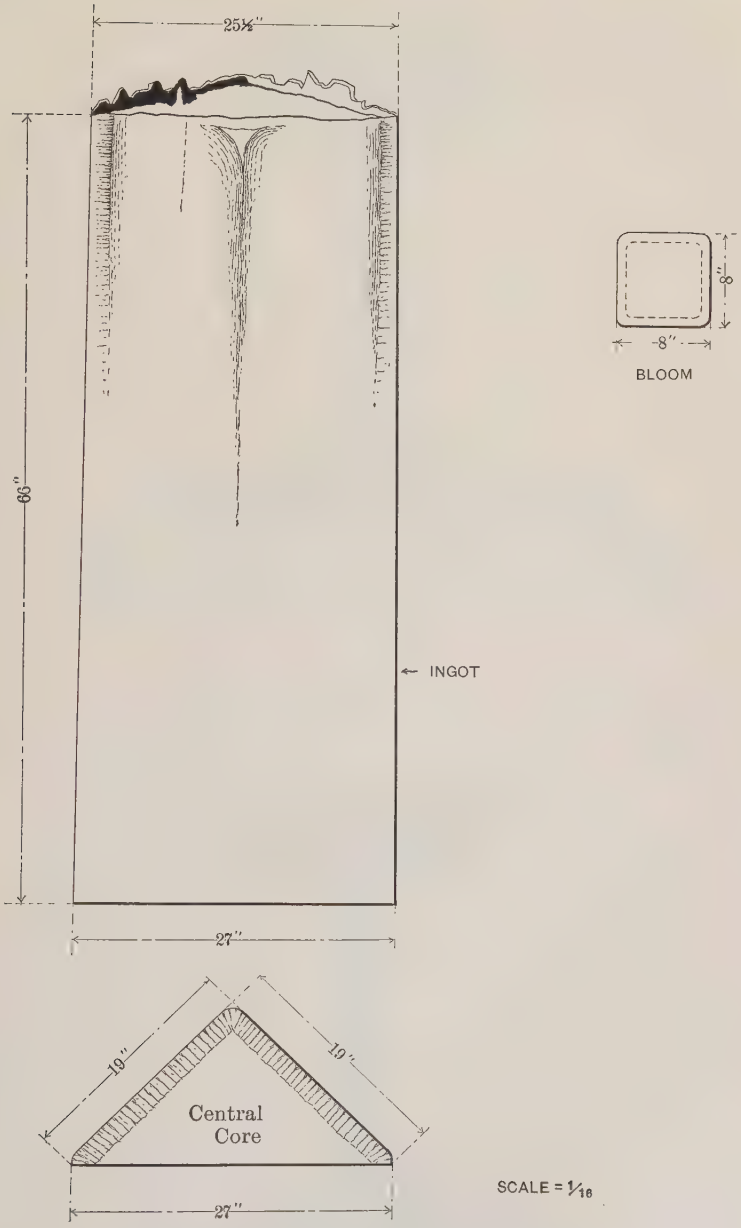
P. H. Dudley, New York, N. Y. (communication to the Secretary*):—Referring to the series of tests on railroad material, which are to be made at the Watertown Arsenal, I can assure you that the New York Central & Hudson River Railroad Co. will be glad to co-operate and take part in furnishing material for tests of different kinds, and also to provide opportunities for service-tests.

I have furnished for the past two decades old and new rails for testing to the U. S. Arsenal at Watertown (see Reports, Ordnance Department, U. S. A.).

Conferring with Mr. Mahon, the sketch, Fig. 2, indicating the ingot-structure after casting and cooling for rails, is offered as a suggestion for one line of investigation.

Ingot-Structure.—The steel in the ingot when allowed to cool completely after teeming, develops more of a shrinkage-cavity than when it goes direct to the heating-furnace, and thence to the blooming- and finishing-trains. The difference in shrinkage in volume of the steel from congelation near 1,300° C. to 21° C. is partly avoided. The columnar structure in the corners of the ingots should be examined for entrained gas, slag, sponginess, and the cavity near the top. The entrained slag in the corners of the ingots often develops as defects in the upper corners of the rails, and the edges of the base. The elongated columnar

* Received Mar. 31, 1908; final revised proof received Apr. 12, 1909.



The central cavity near top depends upon the subsequent cooling and may be small in fair-setting steel, though of magnitude in poor metal.

The etched transverse section of the bloom indicates the zone of vertical blow-holes between the columnar structure and the central core of the upper part of the ingot.

FIG. 2.—PLAN AND VERTICAL SECTION OF INGOT, CUT DIAGONALLY FROM CORNER TO CORNER, INDICATING THE COLUMNAR STRUCTURE.

Segregated metal, split head, developed in center of rail but sound at ends. Carbon in bearing-surface, 0.45; center of head, 0.65; phosphorus, 0.16. Etched 30 minutes.

FIG. 3.—100-LB. SECTION L. S. CO., 1905. FOUR-RAIL LENGTH INGOTS.

Fourth Ave. Tunnel, 3 years and 9 months, lost 15 lb. per yd. Porous steel, flows and wears rapidly. Etched 30 minutes.

FIG. 4.—WORN 100-LB. SECTION C. CO., 1900. FOUR-RAIL LENGTH INGOTS.

structure and central core should be examined from the bottom to the top of the ingot for traces of oxides and slag, and their effects upon the soundness of the metal. The center of the ingot should be examined for pipe, sponginess, and segregation throughout its entire length, and the extent of the portion of the ingot which is unsound. The structure of the entire ingot would show the character of the metal, before it has received mechanical work, and it should be tested for tensile, shearing, and compression strength in several places, particularly where segregation occurs of any of the metalloids.

Transverse tests should be made of the junction of the columnar structure with that of the central core. The blooms would show the ingot-structure reduced before it cooled, and should be examined for soundness and solidity by the same character of tests as the ingot. The specimens should be taken from parts which correspond as nearly as possible to their former location in the ingot. Rolling the metal to the bloom would show the effect of work, and whether the cavities were closed or elongated, and the degree of importance to be attached to their effect upon the steel.

Rails from an ingot of the same heat, numbered from top to bottom, would show the relative solidity and soundness of metal of the finished section, and the increase of physical properties by work and heat-treatment of manufacture.

The ingot, bloom, and rails should all be from ingots in consecutive order from the same heat.

The examination of the physical properties of the metal in the ingot, bloom, and rail should be both in the direction of the length and also at right angles.

The cubic elasticity or elasticity of volume of the metal in the head is of even more importance than the linear elasticity, and should have extensive investigation, for we have little information on this important property.

In case of a failure in compressive tests, note should be made of any types of defects in the metal which conduce to a reduction of the factors of strength.

The metal in the rail-section should be examined in the head, web, and base in relation to its functions as a girder, and then primarily, as to the capacity of the metal in the bearing-surface to receive and sustain the wheel-contacts without undue destruc-

tive deformation of the steel by flow, or of the rail-head as a part of the section. The forward progressive destructive action of the moving-wheel contacts has definite relation to the wheel-load effects, area of contact, and the usual physical properties of the steel, its resistance to repetition of loads, and soundness of the metal.

The slag, gas, and impurities entrained in steel are not uniformly distributed, but localized, by many conditions in the cooling ingot, and manufacture, which are subsequently to be investigated.

Investigation of another series which can be instituted at once is the deformation of the section in inch lengths of the rails cut off in the web just under the fillets.

A force applied in the center of the head by flat and curved surfaces, to represent a wheel-tread, should be used, and the pressures first should have some reference to the wheel-loads, but eventually they should be carried to such pressures as will cause the metal to flow and even shear. The action of the wheel-treads in laterally spreading the metal in the bearing-surface of the head cannot be definitely reproduced in such tests, for they are the results of the application of a great number of loads, and the heads of the rails split or are deformed by detailed fracture.

The ends of the sections should be polished and etched, to indicate the homogeneity and soundness of the metal, before the pressure is applied. They should also be photographed for reference. Then, after destructive pressures are applied, they should be re-photographed to show the disturbance of the metal.

Plungers should be designed to touch only upon the outer corners of the bearing-surface, and see what force is required to crush, split or shear the metal through the sides of the head.

There would be several types of failures in the tests, according to the position of the portion of the rail in the ingot as effected by segregation and slag-inclusions, unsoundness of metal, and formation of the metal in the passes of the rail-trains.

Similar tests could be made upon the base, using the entire section and applying the pressure to the head, and noting what was required to deform or split the base.

Many other tests upon the inch sections are required, which are being formulated in case they are not incorporated in the proposed program.

There must be sufficient metal for tests of adjacent sections, for solidity and soundness of the metal. Some of these tests could be made at once, as the information is desired for current manufacture. It is important to know whether arrangements will be made for furnishing copies of the tests for private information before they are published by the government.

The photograph, Fig. 3, shows a split head in which slag was found a short distance underneath the bearing-surface. This was at the junction of the columnar structure and central core. Fig. 4 shows a section which was porous and lost weight rapidly in the Fourth Avenue tunnel.

J. P. Snow, Boston, Mass.:—Mr. Howard's paper calls for suggestions as to profitable lines of inquiry that may be followed under the program. Recent experience with "crescent" breaks in rails seems to furnish an ideal field for these investigations. The rigid road-beds caused by the unusually cold winter of 1906–07 developed weak rails in unprecedented numbers, and a large majority of the resulting breaks were base-fractures of the crescent type.

In all of the rails that I have examined, the fracture started from a longitudinal flaw in the base, and burst out to the edges of the flange in a sort of conchoidal curve, having a cusp point on one part of the broken base and a corresponding re-entrant angle on the other. Oftentimes one side only of the base breaks out, producing a truly crescent or half-moon-shaped piece. When, however, the rail breaks clear through, the web and head are fractured exactly the same as in square breaks.

The longitudinal flaws from which these breaks commence are of all sizes, from the minutest speck to a visible seam many feet long. They appear to be of two distinct classes: 1, vertical seams with sides in close contact and mostly invisible to the unaided eye; and 2, seams generally oblique, very prominent, and frequently traceable on the base of the rail previous to fracture. The first class I shall call "gas-seams" and the second "rolling-flaws." The seams responsible for the breaks are in the surface of the base, but in the case of "gas-seams"

the body of the metal throughout the base and head is full of these flaws. A fresh break from a very bad rail of this class shows a seamy surface, indicating a cleavage almost as free as a pine board. On a freshly-fractured edge the longitudinal seams show very plainly as silvery patches in the midst of the surrounding gray silky mass of the steel. These seams are described and shown in a paper by Robert Job before the New York Railroad Club.[1] Mr. Job was there discussing nickel-steel, and he attributed the seams to undistributed nickel, but the same defects are found in many cases in carbon steel.

Pieces may be easily broken from the bases of rails affected with either class of these defects, by standing them base up and striking them with a hammer on the side of the base. Sometimes a single blow will break out a crescent, but generally several blows are required. A sound rail cannot easily be broken in this way.

A study of these failed rails has been made at Watertown Arsenal by opening up the metal in a variety of ways and polishing and etching the surface so obtained. I have before you a sample so prepared. It is a piece of a rail that failed from a base-fracture. The head and base have been planed into to different depths, the ends polished, and portions of the base cut out and broken in various ways. The head and base show many white streaks and some dark ones. The white streaks frequently have a fissure in their centers running exactly parallel to the rolling, and in the base and upper part of the head in vertical planes, that is, parallel to the axis of the rolls during the passes of the rails. At different depths of cutting these streaks are seen out of alignment with each other, showing that they occur individually throughout the mass of the metal as independent seams. The dark streaks are found only in the deeper cuts; they are wider and not so definite in outline as the white streaks, contain no manifest fissures, but have been proved to be lines of weakness, and seem to be the softer parts of the metal that lead to crushing and rapid wearing of the head. White streaks are almost indistinguishable in cross-section, while dark streaks show as round spots, blotches and streaks in the section of the head, web and

[1] *Proceedings of the New York Railroad Club*, vol. xvii., p. 514 (1907).

base. They have been illustrated many times by Mr. Job and others.

The second class, or "rolling-flaws," occur only on the surface of the metal. The sides of these seams are blued like mill-scale, they are often strongly fluted, and are generally at an angle more or less inclined to the vertical. Fractures caused by these flaws oftentimes show sound homogeneous metal free from the silvery streaks and specks characteristic of that affected with gas-seams.

In former discussions I have taken the position that the phenomenon here called "gas-seams" was caused by gas-bubbles in the ingots. I am still of the same opinion. The "rolling-flaws" I believe to be due to cracks, tears or folds in the surface of the ingot or bloom.

An interesting fact was developed in a recent examination of a rail badly piped in the head. The pipe extended about two-thirds of the length of the rail. Pieces were broken from the flange with a hammer and but few "gas-seams" were found. More were found in the end of the rail not piped than in the piped end. Where the pipe was the worst it was quite difficult to split the flange. Is there any significance between the presence of the pipe and the absence of gas-seams? If gas-bubbles are the origin of the seams, might not a central pipe, by absorbing the gas in its vicinity, cause a local paucity of these seams? A single instance like this proves nothing, but it indicates a lead the investigation of which may be profitable.

Illustrations of ingot-sections[2] show that gas-holes are commonly found in the steel ingots of commerce. Now, what becomes of these holes during and after rolling? They must be drawn out into seams parallel to the length of the bar and in planes parallel to the axis of the rolls. The gas contained in the holes must be forced into the adjacent metal. This may well affect the chemical composition of the walls of the seam and lead to the white streaks that we find in such abundance in rolled and forged steel.

The sequence between the gas-holes of the illustrations re-

[2] C. L. Huston, Experiments on the Segregation of Steel Ingots in Its Relation to Plate Specifications, *Proceedings of the American Society for Testing Materials*, vol. vi., p. 182 *et seq.* (1906), and H. M. Howe, Piping and Segregation in Steel Ingots, *Trans.*, xxxviii., 3 (1908).

ferred to and the "gas-seams" of our specimens, and between cracks and folds in the skin of ingots and blooms and the "rolling-flaws" in the bases of our rails, seems altogether logical to me, but it may not be satisfactory to all. It would seem to be entirely pertinent to the program laid out at Watertown to try to trace these flaws to their origin.

I hold that the case is fully proved, that crescent breaks are due to these longitudinal seams and flaws; hence, the first step has been taken in the solution of the three-sided problem presented by these breaks, the problem being: first, what defects cause the breaks; second, what generates these defects, and third, what will prevent the said generation.

The remedy for the trouble, whatever it may be, is, as Kipling says, "another story." Openings in low-carbon steels will weld up during rolling much better than those in high-carbon steels. We must, however, have comparatively high-carbon for rails. If gas-holes cannot be welded in this grade of steel, ingots must be cast in such a way that gas-holes will not be formed. Slow teeming at the proper temperature may suffice; sink-heads producing fluid compression may need to be resorted to; recharging with silicon, as advocated by Mr. Sandberg, may help; but it is hoped that some way will soon be found to manufacture rails substantially free from the manifest flaws which are proved to have led to such excessive breakage in the recent past.

Tracing these flaws to their origin and studying a preventive for them seems to me to be a legitimate item of the program announced in the paper before us.

Charles S. Churchill, Roanoke, Va. (communication to the Secretary *):—I have read over with much interest the paper of James E. Howard, Engineer of Tests, Watertown Arsenal, in reference to the program of tests under which current work of the Watertown Testing Laboratory is to be carried on.

The following terse statements are made therein:

"Broadly stated, each grade of steel, chemically considered, can have its tensile properties materially modified by ordinary methods of treatment in common shop use, some grades admitting of wider variations than others, but each admitting of

* Received Feb. 10, 1908.

substantial modifications. . . . So far as known, desired physical properties may be imparted to steel in its final stages by heat or mechanical treatment."

"The working temperatures, reductions in the rolls or under the hammer, and the number of passes in the rolls should be ascertained."

"In addition to questions on the influence of temperature on the final properties, there is also the question of interest as to how much or how little work at a given temperature is essential to accomplish the attainable changes in physical properties."

By way of discussion of this paper and of the subject in general, it seems to me that what is needed specifically just at this time, on the above lines, is more definite information as to the effect of different methods of rolling steel from ingot to bloom and from bloom to finished shape. It would be of great interest to know what are the resulting differences in steel under the following methods of treatment:

1. What are the characteristic differences in blooms of steel rolled from ingots under the following two methods?

A—A reduction of 38.5 *sq. in. per pass in seven passes from ingot to bloom during total time-interval of* 1 *min.* 15 *seconds.*

B—A reduction of 27.7 *sq. in. per pass in thirteen passes from ingot to bloom during a total time-interval of* 2 *min.* 15 *seconds.*

2. What are the resulting characteristic differences in finished steel rolled from blooms to finished shape, the shape being the same size, in the following two cases?

C—A reduction of 7.4 *sq. in. per pass in a total of eleven passes in rolling time of* 1 *min.* 45 *sec., with some delay before final pass sufficient to make the temperature of the finished steel the same as D.*

D—A reduction of 4.7 *sq. in. per pass in a total of eleven passes, in a total period of time of* 3 *min.* 45 *sec., the temperature of the finished product being the same as C, and the finished shape being the same as C.*

But in the case of *D* the blooms have been reheated before rolling, whereas in the case of *C* the bloom was not reheated.

In this investigation it seems to me that one important matter to be determined as far as practicable is the relative amount of welding that has taken place in the segregated portion of the ingot—that is, the upper half thereof—as between processes *A* and *B*.

Henry D. Hibbard, Plainfield, N. J. (communication to the Secretary*):—The fact that the testing department of

* Received Mar. 3, 1908.

Watertown Arsenal, under the direct charge of James E. Howard, is at work with increased means on "tests of steel ingot metal and derivative shapes" is most gratifying to all interested in the subject. The great and excellent work done by Mr. Howard in the past 30 years has added much to our knowledge of the properties of steel and other materials.

To the steel-metallurgist the statement from the program of tests that "a thorough, fundamental inquiry into the physical properties of steel . . . begins with the ingot metal" may seem to be somewhat inadequate, since much depends on the manipulation before casting, which may have a real if not recognizable effect on the steel. It is not easy to get a proper history of the metal before it is congealed in the ingot-mold, nor is it easy to interpret such a history if attainable, and discrepancies between what should reasonably be expected in steel and what is found will still be encountered, though in constantly decreasing frequency.

To illustrate this point, assume the case of a steel containing by analysis 0.02 per cent. of silicon. If the steel has been well made, and the silicon is all present as metallic silicon or as a silicide, all is well. But if the steel has not been well made, some or all of the silicon may exist in the metal as SiO_2. The amount of silicon reported by the chemist then may represent about 0.04 per cent. of SiO_2, or if it exists as a tribasic silicate of iron or manganese then 0.22 per cent. of that compound.

Since the specific gravity of the tribasic silicate is about half that of iron, the 0.22 per cent. by weight will represent about 0.5 per cent. by volume in the steel, which is quite enough to account for much poor quality. Moreover, this 0.5 per cent. may be concentrated along the contact-surfaces between the grains or crystals of the metal in such a way that its effect on the properties is far greater than the proportion it bears to the whole. Chemistry fails us here, though the microscope may help, but if not, the discrepancy existing between the expected properties and those that are found can only be explained by the history of the metal while still liquid. If only a part of the silicon is present as SiO_2, the evil will be less than in the foregoing assumption, but a very little silica or silicate must have a deleterious effect out of all proportion to its relative amount.

The presence of silica and silicates in steel is better under-

stood in Europe than in the United States, if one is to judge from the publications on this subject that have appeared in the technical press.

In order to compensate for the lack of the history of the metal under examination, Mr. Howard proposes that "the material and data should be had from sources where the highest skill now prevails in the art of steel-making." If this means that only the best grades of steel are to be studied there will still be much more to be done with inferior grades. The lessons to be learned from poor steels are of the greatest importance in their educational value.

Still, it may be desirable that the study of the best steels should be undertaken first to establish standards with which steels of poorer qualities may be compared. It is obviously beyond the capacity of any one committee or commission to cover the whole field in one lifetime.

Mr. Howard (communication to the Secretary*):—The members of the Institute have brought out many interesting features and valuable suggestions pertaining to the line of work now being conducted at the Watertown laboratory, and it is desired to express the deep appreciation felt for the aid which has thus been extended. Certain of the suggestions are already being acted upon, the status of the work in the laboratory permitting taking early advantage thereof, while others will be availed of as rapidly as the tests advance.

In research-work of this kind there are fundamental factors which influence the behavior of the material in its subsequent stages of manufacture or use, and it is noticed that due recognition is given such factors in the present discussion, while features of importance in current engineering practice have not been overlooked.

In acknowedging the indebtedness of the laboratory for the encouragement which the Institute is giving its work, advantage will be taken on this occasion to present some recently acquired data on one of the principal derivative shapes of ingot-metal—namely, steel rails. As is well known, a considerable number of rail-fractures have occurred in recent years, and it becomes a matter of deep interest to ascertain, if possible, what

* Received May 5, 1908.

causes have contributed toward these unwelcome results. The severity of service-conditions has increased, without a doubt, and would in itself tend to induce early rupture of the rails, but still this is not all that might do so. The character of the fractures witnessed indicates that certain defects exist in the metal, the presence of which constitutes a menace to the integrity of the steel,

A type of fracture of frequent occurrence is the so-called "moon-shaped" fracture, in which a piece of the base of the rail is detached, as illustrated by Fig. 5. These fractures begin along the middle of their length, following a direction parallel to the axis of the rail for a longer or shorter distance, then changing their course by curving outward, detaching a fragment when the limit of fracture has reached the edge of the flange. The fracture shown in Fig. 5 was made in the testing-machine, on a piece of rail which had developed a moon-shaped fracture in service, the end of the service-fracture being also shown in the cut. A part of such a fractured surface is commonly striated, marking the incipient place of rupture; the balance of the surface presenting a granular appearance radiating from the initial portion. This type of fracture occurs in rails which contain longitudinal streaks, the fractures starting on the line of a streak.

Not infrequently several streaks are present in the same rail, in which case fracture, bending-moments being taken into account, might be expected to occur along the one nearest the middle of the width of the base.

In Fig. 6 are shown the light- and dark-colored streaks in the base of a rail. The metal was planed off to the depth of a few hundredths of an inch, and the surface then etched with tincture of iodine. Surface-creases are sometimes present which have a circular direction, and in shape at least seem attributable to the action of the rolls of the rail-mill. Below such creases streaks are found.

Streaks are encountered in different parts of the rail. Fig. 7 shows longitudinal streaks found in the base of a rail which was planed off at six steps of different depths.

Streaks similar in appearance to those found in the bases are also present in the heads of the rails. Fig. 8 illustrates streaks in the head of a rail at depths ranging from 0.3 to 0.8

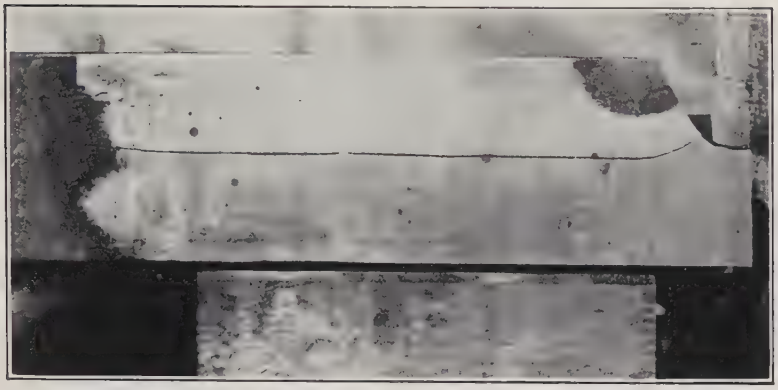


FIG. 5.—"MOON-SHAPED" FRACTURE IN BASE OF A STEEL RAIL.

FIG. 6.—BASE OF STEEL RAIL, SHOWING LIGHT- AND DARK-COLORED LONGITUDINAL STREAKS.

FIG. 7.—BASE OF STEEL RAIL, SHOWING LONGITUDINAL STREAKS FOUND AT DIFFERENT DEPTHS. METAL PLANED OFF AT SIX STEPS.

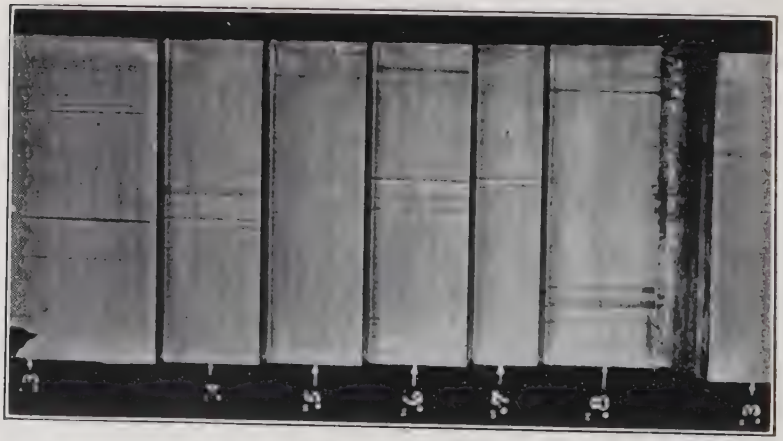


FIG. 8.—HEAD OF STEEL RAIL, SHOWING LONGITUDINAL STREAKS FOUND AT DIFFERENT DEPTHS. METAL PLANED OFF AT SIX STEPS.

FIG. 9.—END VIEW OF STEEL RAIL, POLISHED AND ETCHED, SHOWING MARKINGS ON CROSS-SECTION AND THEIR CONNECTION WITH LONGITUDINAL STREAKS.

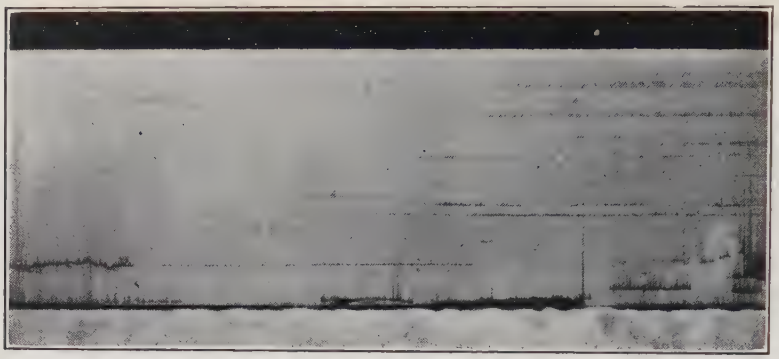


FIG. 10.—HEAD OF STEEL RAIL, PLANED DOWN, SHOWING A SHORT SPLIT ON THE LINE OF A LONGITUDINAL STREAK.

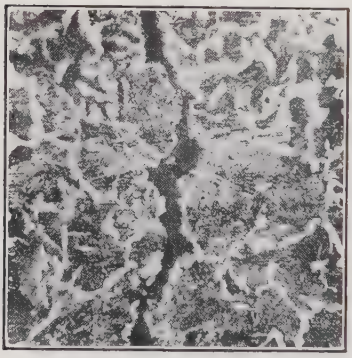


FIG. 11.—Initial State.

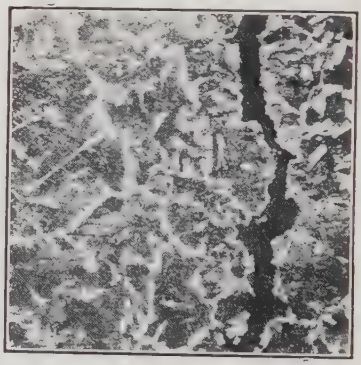


FIG. 12.—After Straining.

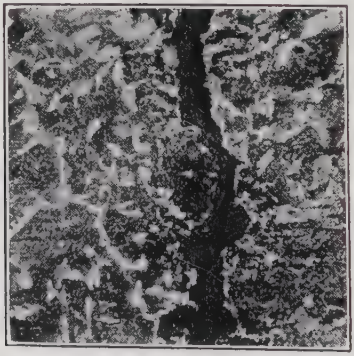


FIG. 13.—After Further Straining.

FIGS. 11 TO 13.—MANGANESE SULPHIDE STREAK IN BASE OF A STEEL RAIL. ALL MAGNIFIED 150 DIAMETERS.

FIG. 14.—PHOTOGRAPH OF FRACTURED SURFACE OF RAIL, AFTER HAVING BEEN RUPTURED IN THE TESTING-MACHINE.

FIG. 15.—CROSS-SECTION OF RAIL, OPPOSITE END OF RAIL SHOWN IN FIG. 14. APPEARANCE AFTER POLISHING AND ETCHING.

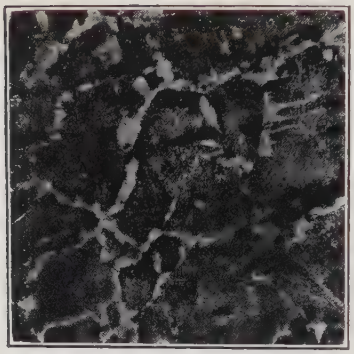


FIG. 16.—Unaffected Part.

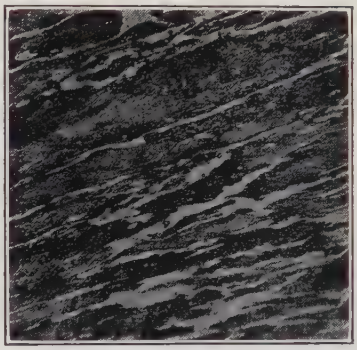


FIG. 17.—Immediately Below Running-Surface.

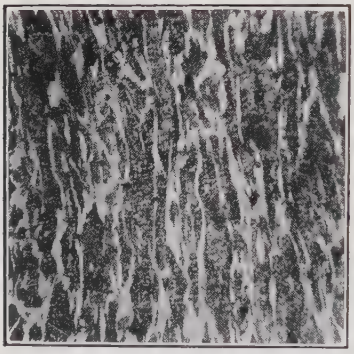


FIG. 18.—Fin on Edge of Head.

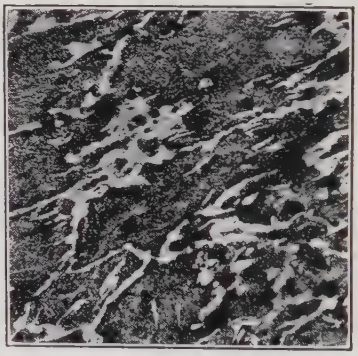


FIG. 19.—At Junction of Unaffected and Distorted Parts.

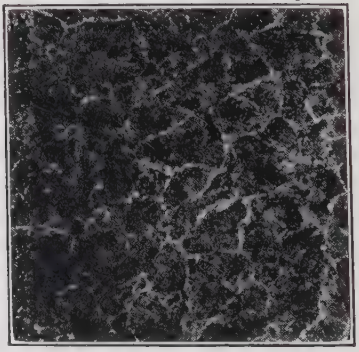


FIG. 20.—Structure after Annealing at Yellow Heat.

FIGS. 16 TO 20.—EFFECT OF WHEEL-PRESSURES ON HEAD OF STEEL RAIL. ALL MAGNIFIED 110 DIAMETERS.

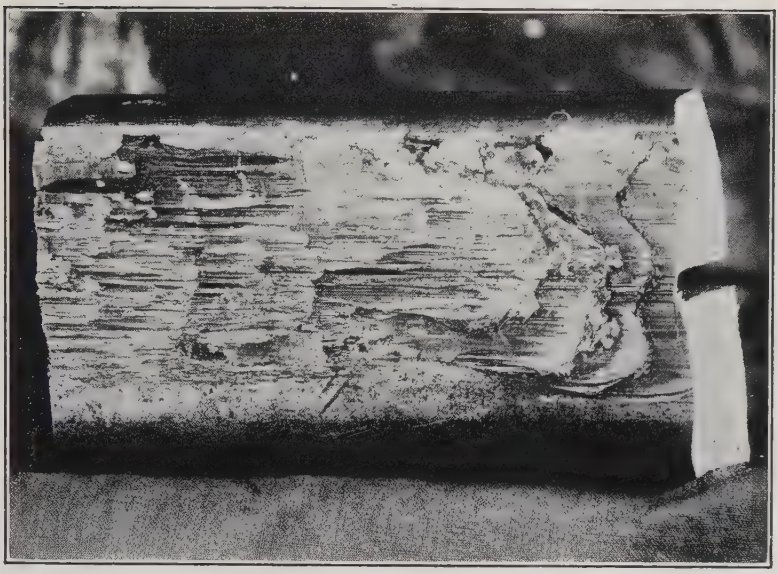


FIG. 21.—RUNNING-SURFACE OF HEAD OF STEEL RAIL, SHOWING ROUGHENED SURFACE CAUSED BY SLIPPING OF WHEELS OF LOCOMOTIVES.

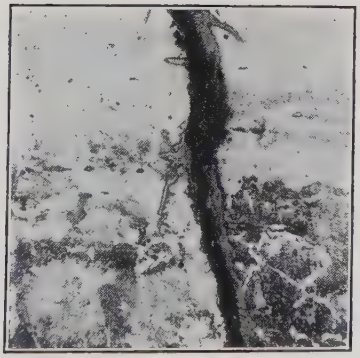


FIG. 22.—THERMAL CRACK IN HEAD OF RAIL SHOWN IN FIG. 21. PART NEXT RUNNING-SURFACE. LONGITUDINAL SECTION OF RAIL-HEAD. MAGNIFICATION, 110 DIAMETERS.

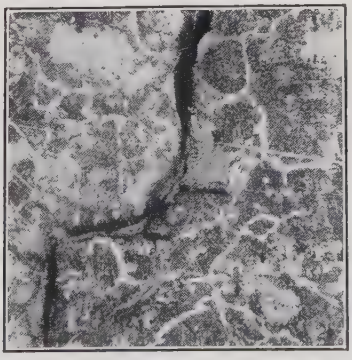


FIG. 23.—THERMAL CRACK IN HEAD OF RAIL SHOWN IN FIG. 21. INTERMEDIATE PART. LONGITUDINAL SECTION OF RAIL-HEAD. MAGNIFICATION, 110 DIAMETERS.

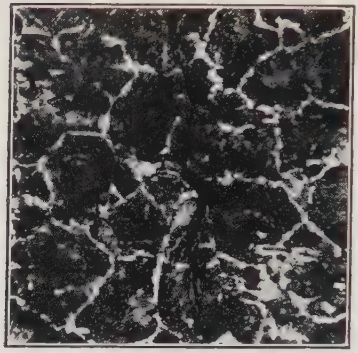


FIG. 24.—THERMAL CRACKIN HEAD OF RAIL SHOWN IN FIG. 21. LOWER PART OF CRACK. LONGITUDINAL SECTION OF RAIL-HEAD. MAGNIFICATION, 110 DIAMETERS.

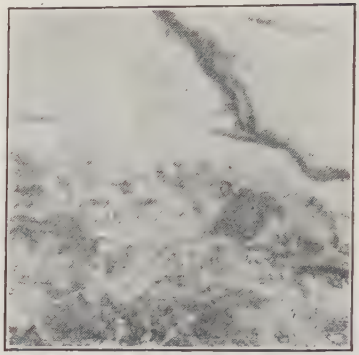


FIG. 25.—A SECOND THERMAL CRACK IN HEAD OF RAIL SHOWN IN FIG. 21, CRACK NOT PENETRATING BEYOND HARDENED METAL OF RUNNING-SURFACE. LONGITUDINAL SECTION OF RAIL-HEAD. MAGNIFICATION, 110 DIAMETERS.

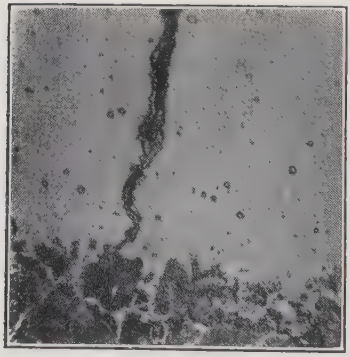


FIG. 26.—THERMAL CRACK IN HEAD OF RAIL SHOWN IN FIG. 21. TRANSVERSE SECTION OF RAIL-HEAD. MAGNIFICATION, 110 DIAMETERS.

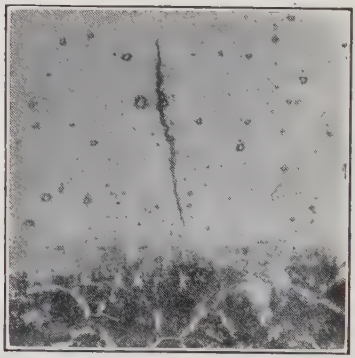


FIG. 27.—THERMAL CRACK IN HEAD OF RAIL SHOWN IN FIG. 21; INTERIOR CRACK IN HARDENED ZONE OF HEAD. TRANSVERSE SECTION OF RAIL-HEAD. MAGNIFICATION, 110 DIAMETERS.

in. below the running-surface. The appearance of the rails illustrated by these photographs may be taken as representative of streaks as they are generally presented to the unaided vision, although characteristic differences are shown when they are magnified.

In Fig. 9 the familiar markings presented on the cross-section of rails are shown, and one streak viewed on the end is seen to extend in a longitudinal direction along the middle of the head. This view is presented to show the connection between the end-markings and the longitudinal streaks.

Fig. 10 is a view of the head of a rail, the running-surface of which has been planed off, and reveals a short longitudinal fissure which is on the line of a dark streak. This fissure was an interior one and did not appear on the surface before planing. It was occasioned apparently by wheel-pressures causing lateral flow of the metal, and which, encountering this plane of weakness, opened a split of short length. This example is taken to illustrate how a split head may result from streaked metal when the streak chances to be near the middle of the width of the head.

From the illustrations which have preceded it would appear that "moon-shaped" fractures in the bases and longitudinal splits in the heads may each, at times, be attributed to streaked metal in the rails.

So common are streaks in certain rolled and forged shapes that it is a matter of deep interest to inquire into the nature of streaked metal. Among the causes which introduce streaks may be mentioned entrained slag, manganese silicate, and manganese sulphide. In collaboration with Dr. Henry Fay, an examination was made of the base of a rail in which there were streaks of manganese sulphide in the steel. Three photomicrographs, Figs. 11, 12, and 13, made during this examination, illustrate the appearance of the streak at three stages, from which the brittle character of the sulphide may be judged, and the readiness with which fractures in the steel may be caused without appreciable display of elongation, by loads applied at right angles to the direction of the streak.

Fig. 11 shows a streak of manganese sulphide in its original state as viewed on the cross-section of the base of the rail. Upon applying a comparatively small bending-stress, bringing

this part of the base into tension, a fissure was caused in the sulphide, as shown by Fig. 12. Further straining of the metal enlarged the fissure, as shown in Fig. 13. This behavior seems to attach to the manganese sulphide properties detrimental to the integrity of the steel, since a fracture once formed in this substance readily extends through the rest of the rail, rupture being completed without the display of appreciable ductility. In the light of previous experience, manganese sulphide seems the most serious one of the three causes of streaks previously enumerated in promoting brittleness of fracture. This also is an interesting feature of the case, since some of the rails, notably those of early English manufacture which are very streaked, are reported to give good service in the track, even although of lighter weight than rails with which they are compared, these rails having streaks due to the presence of manganese silicate.

Fig. 14 is a view of the surface of the rail fractured in the testing-machine. The rail was loaded transversely, with the head on the tension side of the bend. Fracture began at the edge of the running-surface where a fin of metal had been formed by the flow of the metal under the wheel-pressures. Attention is invited to the radiant appearance of the fractured surface, the center of which marks the initial point of rupture. This appearance is generally so well defined in steel that the development of a fracture may be traced to its origin, a source of material advantage not infrequently when judging of the causes of rupture.

Fig. 15 is the opposite end of the rail shown by Fig. 14, which was polished and etched. The markings on the surface are less pronounced than those usually displayed by rails. The shape of the head in cross-section suggests the flow of metal which took place when the rail was in the track. This flow necessarily produces a distortion of the microstructure, which effect extends to a depth of several hundredths of an inch below the running-surface. Impaired ductility characterizes the affected zone. Figs. 16 to 19 show the structure of the metal in the above rail. Fig. 16 is the normal, granular structure of the head; Fig. 17 shows the distorted shape of the grain immediately below the running-surface, at places along the middle of the width of the head; Fig. 18 shows

the shape of the grain at the edge of the head—the metal composing the fin. At this place the metal has flowed so as to bring the longitudinal axis of the distorted structure into vertical position; Fig. 19 shows the structure at the junction of the distorted and the unaffected metal, the depth generally ranging from 0.03 to 0.06 in. below the running-surface.

Fig. 20 shows the subsequent structure of this rail after it had been raised to a yellow heat; the same structure pertained to the metal immediately below the running-surface as elsewhere in the head after this annealing had taken place. With this equalization of structure there was effected a restoration of the ductility of the steel, and the rail then displayed good bending-qualities with either the head or the base on the tension side.

The slipping of the wheels on the rail, such a movement as occasionally takes place with the driving-wheels of the locomotive when starting, in addition to roughening the surface intensely heats the metal. The appearance of the running-surface of a rail thus roughened is shown by Fig. 21, reproduced from an illustration in *Tests of Metals*, 1906. While a loss in ductility accompanies the flow of the surface-metal, a more serious result, which affects the integrity of the steel, results from the superficial heating of the head of the rail; the sudden cooling of the heated metal by conductivity of the cold metal below hardens the surface by its quenching effect, and, at the same time, the surface-metal is thrown into a state of tension. Cracks are, in this way, formed in the running-surface, which have a tendency to extend deeper into the metal, and may eventually end in complete fracture of the rail.

Figs. 22, 23, and 24 show different parts of the same thermal crack, one which had been made in the rail illustrated by Fig. 21. This crack appeared on the surface of a longitudinal section of the rail-head. Fig. 22 shows hardened metal at the immediate surface of the rail-head. This portion of the metal remained unetched by the reagent used, a 4 per cent. solution of picric acid. The inclination of the crack will be noticed, indicating the direction of the flow of the metal, preceding, doubtless, the hardened period. Figs. 22, 23, and 24 show this crack at successive depths.

Fig. 25 shows another thermal crack of lesser depth, one

which has not extended through the zone of hardened metal. This crack was also found on the longitudinal section of the rail-head.

Figs. 26 and 27 represent thermal cracks viewed on a surface of the head, on a transverse section of the rail. These cracks therefore extend, in one direction, parallel to the length of the rail. The smaller crack of the two here illustrated was an interior one, which formed within the zone of the hardened steel.

It may be remarked that Dr. Fay found manganese sulphide on the walls of many thermal cracks of this rail-head. This result would seem to show that the sulphide caused lines of lower resistance in steel as regards both stresses applied to the cold rail and strains accompanying the changes in temperature during the hardening of the metal of the running-surface.

The Coal-Briquette Plant at Bankhead, Alberta, Canada.

Discussion of the paper of Edward W. Parker, p. 236.

William H. Blauvelt, Syracuse, N. Y.:—Is the coal itself from which the briquettes are made of good quality for steaming-purposes?

Mr. Parker:—It is an anthracite coal mined near Bankhead and used on the Canadian Pacific railroad. It contains about 83 of carbon, 8 of ash, and 8 per cent. of volatile material. With regard to briquetting of a mixture of fine coal and coke-dust, I understand that a plant has been installed near Detroit in which coke is mixed with anthracite; also the plant of the United Gas Improvement Co. near Philadelphia is briquetting a mixture of anthracite coal, the resultant briquettes being used in the gas-retorts.

Mr. Blauvelt:—The purpose of the Detroit briquette-plant of the Solvay Process Co. has been partly to use up the coke-breeze from the by-product ovens and partly to utilize the pitch. There is no necessity for mixing anthracite with the coke-breeze, but the supply of breeze is limited, and anthracite

is added in order to increase the output. Soft coal also is mixed with the anthracite and coke; but in the practice at Detroit it has been found that unless the briquetting is very well done the addition of much soft coal makes a smoky briquette. If used by consumers accustomed to anthracite, it is desirable to make a briquette which is almost smokeless.

The problems encountered in suiting briquette-manufacture to American conditions are very different from those in European practice. American coals are relatively low-priced and of good quality, so that in most manufacturing districts, at least in the East, the natural supply of coal is well adapted to the requirements of the manufacturers. Anthracite coal as prepared for domestic consumption is an almost ideal fuel. In Europe, on the other hand, coal is high-priced and very many deposits are non-coking, so that the fine coal can only be made available for use by forming it into briquettes.

There are three different fields in the United States for the development of the briquette-industry :

1. In the East, where the domestic trade is practically supplied by prepared sizes of anthracite, and where steam-coals are of excellent quality and relatively low in price. In this field coal-briquettes offer a substitute which can be sold at a lower price than anthracite. Moreover, there are also a number of special purposes for which briquettes are adapted.

2. In the middle West, in regions like the Hocking valley, where the conditions are similar to those in parts of Europe, the coals are non-coking, and the fine coal, as such, is to a large extent worthless. If formed into briquettes, however, it makes an excellent fuel.

3. In the far West, where there are large deposits of lignite. Many of these lignites crumble into a coarse powder after an exposure of a week or two to atmospheric action, but if briquetted they form an excellent fuel, and withstand indefinitely atmospheric action.

Many briquette-enterprises have been started in the United States, but only a few have met with success. In some cases failure has been due to improper methods of manufacture, but oftener it has been caused by a neglect to give proper consideration to the commercial conditions; either the available raw material was not adequate or suitable, or else the process

selected did not produce a briquette suited to the local market-conditions.

The manufacture of briquettes may be divided into two classes: large briquettes, and small briquettes, or eggettes. The former weigh from 5 oz. to 20 lb. each, the larger size being about the weight of the largest briquettes made in Europe; this class is suitable mainly for industrial purposes, and can best be made on presses of the reciprocating type. The small briquettes, or eggettes, usually weighing from 1.5 to 3 oz., are best made on a rotary press of the Belgian type. If an attempt is made to produce the smaller briquettes on reciprocating presses, the output is so reduced as to make the operation commercially impracticable. On the other hand, rotary presses are not adaptable to briquettes weighing more than 5 or 6 oz. each.

James Douglas, New York, N. Y.:—The only enterprise in the West that I know of that has made briquettes is the Arizona Copper Co., which installed a plant in order to utilize the refuse from its large coke-piles; and though not in operation at the present time, it saved the company from a shut-down two years ago when coke was very scarce. I understand that the total cost of the briquettes at that time, including the tar for binder, which had to be imported, amounted to about $1.50 per ton.

With regard to briquetting lignites, I think the trouble would be to secure a proper supply of tar for the binder. At present, tar is extremely expensive in the lignite-fields. If by-product coke-ovens were introduced for coking bituminous coal, the heavier products from the distillation of the tar would then be available for a binder. In addition, the creosote would be extremely valuable for use as a timber-preservative. To my mind, the most important question at present in the far West is the introduction of by-product coke-ovens to supply creosote for timber-preservation, tar as a briquette-binder, ammonium sulphate as a fertilizer, in making which the sulphur going to waste from our furnaces would be utilized. If the farmers would only use artificial manures before their land is actually impoverished, they would not have to sue the smelters for damage to their crops, for the sulphur would actually nourish instead of destroying plant-life, when converted from noxious gas into one of the ingredients for artificial fertilizer.

Mr. Blauvelt:—One of the reasons why the by-product oven has not been introduced into the far West is the question of the market for the by-products. It is possible that the briquette-industry and the by-product oven may be developed together in that region, the briquette-plant consuming the pitch from the tar, and the ammonium sulphate being used as a fertilizer.

The cost of manufacturing briquettes, of course, varies greatly with local conditions, the size of plant, etc. For a plant of moderate size the operating-cost might be from 50 to 75 cents a ton, not including the binder. In large plants these figures should be materially reduced. Some of the prominent manufacturers of pitch, which is recognized as the principal binder for briquettes, have appreciated the fact that if the briquette-industry is to grow to considerable proportions in the United States, the question of pitch for the binder must have special consideration. This condition may develop the use of a special hard pitch which can be shipped in bulk under a low freight classification, and avoid the expense of barrels or other packages.

Dr. Douglas:—I have more doubt about the ammonia than about the tar. The Santa Fé railroad is the only one that is importing creosote in large quantities. Creosote, of a value of 3 cents per gal. in Europe, costs the Santa Fé road 8 cents a gal., yet, in spite of this fact, it is preferred to zinc sulphate. If coke-oven by-products could be used with any advantage in the West, I do not think there would be any difficulty in getting rid of the tar, but the farmers have not yet been taught to use ammonium sulphate as a fertilizer.

C. G. Atwater, New York, N. Y.:—Dr. Douglas has suggested that if a plant of by-product ovens were erected in the far Western States there would be no difficulty in disposing of the tar in the form of pitch for briquette-making, but that a market for ammonia would be doubtful. It seems to me that under present circumstances no difficulty need be anticipated. It is merely a question of extending the plant. If ammonium sulphate is made, a market could be found for it as a fertilizer. During 1907, Japan alone imported 64,000 tons of ammonium

sulphate from England, paying freight around Cape Horn or through the Suez canal, so that there should be no difficulty in disposing in Japan or elsewhere of the 2,000 or 3,000 tons that a plant of moderate or large size would produce. The conversion of ammonia into sulphate, however, requires a supply of sulphuric acid to the extent of about 1.25 tons of chamber acid to a ton of finished sulphate. A plant for making the acid could be erected at the ovens, as has been done in several cases elsewhere, or the acid could be obtained from some one of the existing acid-works on the Pacific coast, if freights were favorable; or, as is frequently done in Germany, the ammonia could be concentrated to strong crude liquor to lessen the shipping-weight, and transported to a Pacific coast port where acid could be obtained at a favorable cost, the transformation to sulphate being made at that port.

There is good reason to believe that an investigation of the matter from a commercial point of view, in connection with a prospective plant of by-product ovens, would disclose a favorable field for the disposal of ammonia on the lines above mentioned.

The sale of the ammonia from a plant in the far West has seemed hitherto an easier matter than the creation of an outlet for the tar. The tar, from this point of view, was the obstacle to the introduction of the by-product oven in this territory. But the growth of the briquette-industry and the confidence that has been expressed at this meeting of the Institute that this growth is on a stable foundation and liable to receive considerable extension in the near future, would effectually remove this obstacle. With a good prospect for the advantageous disposal of these two important by-products, it would seem probable that the installation of by-product ovens to treat the coals of the far Western mines is not very distant. Concerning the demand for coke, there is reason to believe that the market will easily absorb more than it has ever yet received.

The Bogoslovsk Mining Estate.

Discussion of the Paper of William H. Shockley, p. 274.

H. W. Mussen, Collingwood, Ontario, Can. (communication to the Secretary*):—Doubtless all engineers who have paid more than a casual visit to Russia have come into contact with that formidable document, the "*smieta*," or estimate of the Russian manager. It is an institution which seems to be inseparable from Russian mining-methods. As Mr. Shockley points out, the year's costs "agree very closely" with the figures of the *smieta* made at the beginning of the year. In cases which have come under my notice, I have been satisfied that there had been a good deal of "rigging" of accounts to this end. The sum spent by the manager was carefully kept down to the total of the estimate, because it was well known that no more money would be forthcoming, and when this is done it is not a difficult matter to see that the right amounts come under the right heads in the final accounts. I do not say that this is always the case, but I have come across notable instances. Looking at it from the Russian owner's point of view, I think that the balance of the argument is usually in favor of the *smieta* as limiting "impulsive indiscretion" on the part of managers.

The Russian Mining Code of to-day leaves much to be desired, but this is thoroughly appreciated by the official world there, and great changes are spoken of for the future. In many, if not in most, cases the laws are interpreted liberally, and a great deal is left to the discretion of the district mining engineer or inspector, with whom it is necessary to keep in close touch.

As Mr. Shockley points out, the officials of a mine, who are always Russians, are held personally responsible for accidents resulting to employees, quite independently of any compensation which may be made to the sufferer. This seems startling enough, but I have always found that a penalty is imposed only

* Received Apr. 2, 1908.

where gross negligence on the part of the company's officials is proven in court. A great many cases go to trial, but very few result in punishment.

The great number of holidays kept by Russian workmen is, indeed, a trial to one who is trying to push on work during a short summer season. Each mine or company working has prepared for it an official list of days upon which work may not be demanded of the men; but this does not apply to certain forms of employment, such as blast-furnace work, pumping, etc., which, from their nature, must be carried on continuously. In such cases it is provided that, if the shift is more than 8 hr. long, 4 days in each month must be given as rest-days. If the shift is 8 hr. or less, full time may be demanded each month. It is usual in Russia to pay 50 per cent. additional wage for working on legal holidays, and this goes a great way towards keeping the work going steadily.

Mr. Shockley's paper is intensely interesting and valuable to any one operating in Russia, as the Bogoslovsk Estate is one of the most important mining-enterprises of the Empire.

Dip and Pitch.

Discussion of the paper of Dr. R. W. Raymond, p. 326.

R. W. Raymond, New York, N. Y.:—Since the presentation of my note on this subject at the New York Meeting, Professor Louis has pointed out an error in my statement of his conception of "pitch"—namely, he understands the pitch, in the restricted sense which he proposes, to be the angle of inclination of the axis of an ore-body, measured on the vertical plane drawn through the body; whereas, I represented him as proposing to define "pitch" as the angle of inclination of an oblique body, measured on the plane of the dip, in the direction of the strike. The difference will appear upon consideration of the accompanying isometric projection, Fig. 1, in which $ABCD$ represents the plane of the inclosing ore-deposit (vein or bed), and AC the axis of the oblique ore-shoot. ABE is the angle of the dip, as universally understood; and ACE

is, according to Professor Louis's proposal, the angle of the pitch, which is measured in the vertical plane *ACE*. But, in my restatement of his proposal, I made *ACB*, measured in the plane *ABCD*, to be the pitch. Since his original communication contained this diagram, there is no excuse for my misunderstanding of it. But there is an explanation, which seems to me more important than an excuse—namely, without carefully studying Professor Louis's illustrative diagram, I hastily assumed that his conception of "pitch" was that which I had myself entertained for many years, and believed to be entertained by American mining engineers generally.

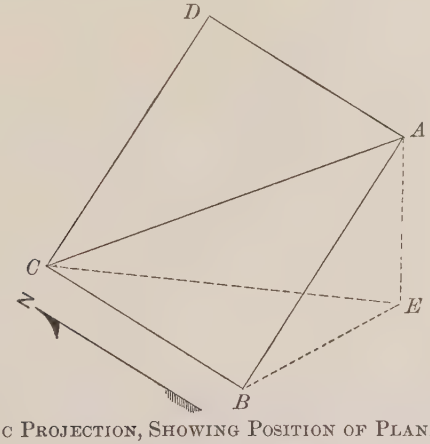


FIG. 1.—ISOMETRIC PROJECTION, SHOWING POSITION OF PLANE OF BED OR VEIN, AND AXIS OF ORE-BODY THEREIN. *ABCD*, plane of vein; *AC*, axis of body; *AE*, vertical; *BC*, *CE* and *BE*, horizontal.

Was I wrong as to that usage? I hope the members of the Institute will help me to decide this question. In recalling the sources of my own impression, I realize that in American practice I have seldom or never heard of "pitch" as an angle distinct from the dip, and independently determined by survey. Our mine-surveyors recognize, but do not employ, the old usage of "dip" and "pitch" interchangeably; and I think they generally apply the term "pitch" to the case named by Professor Louis at the beginning of his communication, as follows:

"It is well known that the ore-shoots in a mineral vein do not, as a rule, coincide with the dip of the vein, but make an angle with it; in the same way, lenticular masses often are oblique to the direction of maximum dip of the plane in which they lie. We have no word to designate this obliquity in either case."

According to my view of American practice, the direction of the pitch is usually stated in terms of the strike. Thus, if a vein strikes N. and S., an elongated oblique ore-body in it is usually said to pitch north or south in the vein, whatever be the dip. According to Professor Louis's definition, the pitch would have a new strike of its own; and an ore-body could not pitch north or south in a N–S. vein unless the latter were vertical. Moreover, the American usage, as I understand it, furnishes exactly what Professor Louis says is needed—namely, a measure of the "obliquity" of a lenticular mass "to the direction of the maximum dip of the plane" in which it lies. The line of maximum dip and the line of the axis of the oblique body both lie in the plane of the dip; hence, it seems to me, the angle between them is to be directly measured in that plane.

Moreover, this method of estimating pitch is the most convenient for the practical miner. The pitch of the ore-body determines how far he must run each successive level to reach it; and, since he runs these levels on the plane of the dip, his guiding measurements should be made with reference to that plane.

Again, the inclination of an ore-body, on the plane of the dip, "to the direction of maximum dip" is a thing the miner wants to know, and to use daily. If we do not call it the pitch, then we must have another name for it. But the exact course and inclination which Professor Louis wishes to call "pitch" is something that the miner does not thus need and use. The course and dip of the vein or bed having been determined, he naturally refers to that plane all other directions, dimensions, and angles involved in the work of mining. The whole mine appears to his thought like a picture drawn upon that plane. In fact, the mine-maps often include a profile, representing the workings on the plane of the vein, which is for many purposes more convenient than a projection of them upon a vertical plane. In presenting mining cases to judges and juries, I have often found the "profile" to be more easily understood than the vertical section and projection.

Perhaps my greatest objection to Professor Louis's "pitch" is, that it introduces a new course, whereas mine retains only the course of the vein. In a private letter, Professor Louis defends his definition (referring to the diagram given above) as follows:

"I call *ACE* the angle of pitch, because it has a real physical meaning; it is the actual angle of inclination of the line *AC* (the axis of the ore-shoot) to the horizontal, and no other angle is. If I want to sink a shaft to follow the axis of the ore-body, and, needless to say, that is what I do want, I must set out my shaft at the angle *ACE*, and of course in the azimuth of the line *EC*. Furthermore, *ACE* is easily measured in practice, while it would be rather difficult to measure the angle *ACB*."

To this I reply that in my experience it has been very rare that inclined shafts have been sunk to follow the axes of inclined ore-bodies. The ordinary practice is to sink either vertical shafts, outside of the vein, or inclined shafts within it, following the dip. In either case, the levels are run on the strike, and the ore-bodies are cut by the levels. Professor Louis overlooks the circumstance that several ore-bodies, pitching at different angles (and sometimes, though seldom) in different directions, often occur in the same mine. It is not usual to develop these by separate shafts, following their axes.

Nor do I see that it is easier in practice to measure *ACE* than *ACB*. In fact, however, the pitch *ACB* is usually first stated qualitatively, not quantitatively; and the mine-development afterwards measures it. The commonest way of stating it is not in degrees and minutes. For instance, the miner says his vein strikes north and south, and dips east, while his ore-body pitches north or south so many feet to the hundred. If he were told that the said ore-body really "pitched" N. 39° 42′ E., he would not understand why this odd new element had been introduced into the problem which was so simple without it.

Prof. Henry Louis, Newcastle-on-Tyne, England (communication*):—Dr. Raymond's interesting note and discussion furnish an admirable example—if such were indeed needed—to prove how far mining still is from being a science. The fundamental need of an exact science is precision in definitions; and here we see that Dr. Raymond and myself are proposing to define differently a word, in common use among miners, which each of us has used scores of times without suspecting that the other could understand thereby anything different from his own conception of it.

There is, of course, no difference of principle involved. It is merely a question of convenience, whether it is better to

* Received June 11, 1908.

measure the angle of pitch in the plane of the vein or ore-body, or in the vertical plane in which it lies. I prefer the latter method for various reasons.

In the first place, it is more scientific, because when we speak of the angle of inclination of any straight line to the horizontal, we always mean the angle as measured in a vertical plane, and there seems to be no reason why we should make an exception in this particular case; furthermore, by following this general geometrical practice, there is no need to state more than this one angle; whereas, if we adopt Dr. Raymond's method we must state not only his angle of pitch, but the angle of dip of the plane in which it is measured, before we know what the real inclination to the horizontal is. I admit that in the majority of cases it makes no practical difference, because the plane of the vein is so nearly vertical, and the pitch also is generally rather steep; and further, as Dr. Raymond says, we usually speak of it in somewhat lax terms. I admit also, that, as he says, "if a vein strikes N. and S., an elongated oblique ore-body in it is usually said to pitch north or south in the vein, whatever be the dip;" but in saying this, I understand that the ore-body pitches northwards or southwards, as the case may be, and not that its azimuth is due north or south; I cannot agree that my definition postulates a new strike for the pitch. The direction of strike, which is the azimuth of a horizontal line in the ore-body, is not affected at all: and my method affords an easy means of determining the azimuth of the ore-body, in the few cases where this is required to be determined, while this is less easy to calculate from Dr. Raymond's definition. It is, of course, quite usual, as Dr. Raymond says, for the pitch to be measured not in degrees, but by the number of feet of horizontal displacement to a given height; I should fear that Dr. Raymond's method would lead miners to speak of the height between two levels not in terms of true vertical height, but in terms of distance apart measured along the plane of the vein.

To be quite fair, I may perhaps be allowed to digress sufficiently to explain that for many years past I have never used projections on the plane of the vein, but always horizontal and vertical projections; *e. g.*, in measuring up ore-bodies, I always use for veins: horizontal thickness into vertical height into length along strike, and for beds: area in plan into vertical

thickness, and I feel sure that the engineer who has given this system a trial will never revert to the old method of employing true thickness instead of horizontal or vertical thickness, as the case may be. This practice has caused me to avoid projections on the plane of the vein, and I find that it simplifies my surveying work immensely. I have already said that in the majority of cases, where mineral veins are concerned, it does not matter very greatly which definition of pitch we work to. But in the case of oblique lenticular masses, which we so often have to deal with in European mining-practice, the plane in which these lie is often comparatively flat and their pitch is often considerable; in these cases, it is usual, and is perhaps the best practice, to develop the ore-bodies by means of an inclined shaft along or parallel to the axis, and in that event my angle of pitch is the angle of inclination at which the shaft has to be sunk, while Dr. Raymond's is not, although the former can, of course, be calculated from the latter, if the dip of the plane of the ore-body be also known.

In proof of the real practical importance of the knowledge of this angle of pitch, I may say that last summer, in the course of my regular professional work, I came across two instances, within three months, of mines where such lenticular ore-bodies had been lost, and had to be picked up by drifts, because the shafts had been laid out to follow the angle of dip instead of the angle of pitch, as defined by me.

I repeat, however, that I do not think that it matters very greatly which definition of pitch we adopt; the essential thing is that we shall definitely decide upon one, and keep to it.

Dr. Raymond:—I am heartily grateful to my friend, Professor Louis, for his candid and lucid criticism, which, however, serves rather to define and emphasize the difference between us, than to adduce new considerations, calling for rejoinder from me. In fact, I could not contradict Professor Louis's statements, because they are true; and, as to one of the two questions involved in this discussion, they must be weighed, not thrown out.

The two questions to which I refer are these: (1) which of the two definitions of "pitch" is theoretically the better—that is, which would be preferable if the term were a new one, and we were entirely free to fix its meaning? and (2) which of

them is now employed by the larger number of intelligent English and American mining engineers?

It is evident that if Professor Louis's definition of "pitch" were accepted, what I call "pitch" would need a new name—and *vice versâ;* and it is probable that the question which pitch should hereafter be the standard pitch would be largely affected by the extent of the present intelligent usage. It is much easier and wiser to introduce a new name for a new conception than to introduce a new meaning for an old and familiar name. I hope, therefore, that engineers in both countries will freely express their views and report their practice in this respect.

After 45 years of professional practice, I am convinced that it is much better to have such questions thoroughly discussed than arbitrarily and prematurely decided. If the discussion of this particular question should last a long time, all the better. Engineers can easily avoid all embarrassment to themselves or others by simply stating, when they speak of "pitch," in what plane the pitch they mean is measured.

Edward H. Williams, Jr., Woodstock, Vt. (communication*):—I agree with Dr. Raymond in the statement that "dip" and "pitch" are interchangeably used. This is generally the case in our American anthracite-region, where "a dirty pitch" is the term applied to a dip between the angle at which a car can be somehow hauled to the face by a mule, and the higher angle, at which coal will slide down on a sheet-iron bottom. In the diagram given in Dr. Raymond's discussion, AB, according to this usage, would be the angle of the "dip" or "pitch," and the line AC would be called a "slant," forming the angle BAC to the right on the dip—the observer being supposed to stand with both feet on the strike, and look down the "dip" or "pitch." In the anthracite-region, we drive "slant" roads at any or all angles which will permit us to haul up the cars with mules, or to "switch-back" loaded cars from one level to another. Eckley B. Coxe originally ran all his "empties" to the slopes by "slants," and frequently started the breasts from the slants. I have carried this idea to the case under consideration, and would still call AC a "slant."

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* Received Aug. 1, 1908.

Dr. Raymond:—Professor Williams, representing the usage in our anthracite-regions, recognizes no distinction between "dip" and "pitch." That this should be the case in coal-mining is not surprising, because coal-seams do not present the case stated by Professor Louis, of a body lying in the plane of a vein or bed, with an axis oblique to the dip. Such an axis would be what Professor Williams calls a "slant;" and it is noteworthy that he would determine the angle of the "slant" in the plane of the coal-seam, which is the way in which I would determine the angle of the pitch, if that term were applied to the position of AC as the axis of an ore-body.

Algernon Del Mar, Pasadena, Cal. (communication*):—Dr. Raymond's statement represents, I think, the general practice in the West—the "dip" being the inclination of a vein at right angles to the strike, and the "pitch" the inclination of an ore-shoot in the plane of the dip.

W. Spencer Hutchinson, Boston, Mass. (communication†):—I have been familiar with the use of the word "pitch" for many years, and it has never occurred to me to attach to it any other meaning than the conception defined by Dr. Raymond. I have, however, for the past 14 years used "rake" whenever I wished to refer to the angle made by the axis of an ore-shoot in the plane of the vein. Although the term is not original with me, I am unable to recall where I first heard it used, and I am not aware of any general use of the term. It appeals to me, however, as being distinctive and preferable to "pitch," which is often used loosely and sometimes interchangeably with "dip."

There is, however, no question of the advantage pointed out by Professor Louis of sometimes laying out a shaft in such a way that it will lie in the azimuth of the ore-shoot and be parallel to it. I once had occasion to do that myself with an ore-shoot having a rake of 30°, and I have known other cases where great economy might have been effected if that plan of development had been followed.

Dr. Raymond:—Mr. Hutchinson's suggestion of the term "rake," instead of "pitch," is a subject for separate considera-

* Received Aug. 18, 1908. † Received July 31, 1908.

tion. The main present question, as I understand it, is whether the angle of the oblique axis of an ore-body within a vein or bed ("rake" or "pitch," or whatever else it may be called) should be measured in the plane of the vein or bed. As to this point, if I correctly understand Mr. Hutchinson, he agrees with my view.

C. T. Mixer, Salt Lake, Utah (communication*):—I have been in the habit of using "rake" as well as "pitch," in referring to the inclination of an ore-shoot, generally regarding "rake" as representing the definition of Professor Louis, and "pitch" as expressing that of Dr. Raymond. For instance, I might describe the "rake" of an ore-shoot descending southwards in a N–S. vein dipping west, as south so many degrees west, and so many degrees of declination below horizontal; but I might advise the miner to drift south from his shaft on each lower level, because the "pitch" was south.

I think we should adopt both of these terms in our mining vocabulary; and, since the International dictionary defines "rake" as the inclination from a perpendicular, I am inclined to believe that the two words should be distinguished as I have explained above, though some may regard the reverse as preferable. In fact, Professor Louis's definition may require two terms, one for the course, and the other for the dip, of the axis.

With regard to Professor Louis's method of measuring ore-bodies by the use of horizontal and vertical projections, there is no doubt that this gives the correct estimate with the least work. An objection to the method is, that it tends to give a wrong conception of the true width of a vein, which, from the standpoint of economy of operation, it is necessary to keep in mind. A vein dipping flatly might be so narrow that it could not be mined economically without removing part of the wall-matter, and yet, by the study of horizontal-projections, one might unconsciously be led to think that it was wide enough to be mined without removing the wall-material. Again, by mapping on the plane of the vein we are kept posted as to the actual stoping-distance that we will have upwards, while a vertical projection might mislead us.

I trust that all these matters will be thoroughly discussed by

* Received Aug. 6, 1908.

members of the Institute, so that we can arrive at more uniform methods and terms, which is certainly to be desired.

Dr. Raymond:—Mr. Mixer proposes "rake," not as a substitute for "pitch," but as a companion term, designating what Professor Louis has proposed to call "pitch." As to the latter, he apparently confirms my view. Concerning his citation of the dictionary-definition of "rake," it may be remarked that this definition does not fully support him; for it makes "rake" a departure from the perpendicular, whereas Mr. Mixer's statement of his own practice shows that he measures it from the horizontal. As to that matter, I think that he, together with all other mining engineers, will heartily agree with me that we ought to get rid of the perpendicular as a basis of departure for the measurement of angles—or, to speak more accurately, that when neither horizontal nor perpendicular is named, the former should always be understood. Vagueness in this respect is frequent, and should be unpardonable.

Victor G. Hills, Denver, Colo. (communication*):—On this subject, I regard the view of Dr. Raymond as correct, both theoretically and practically.

1. Theoretically, it seems to me that when the angle of the axis of the ore-body is to be measured in a vertical plane, as proposed by Professor Louis, the word "dip" is the proper term. Then, in a case which involved an ore-shoot as distinguished from the inclosing vein, we would say, "the dip of the vein" or "the dip of the ore-shoot." If it should be objected that the term "dip" should be more carefully guarded and confined, I still see no objection to using it always when we mean the angle made in a vertical plane from the horizontal; and I believe that it is generally so used. There is no possible confusion, since it is always accompanied by an explanatory word or phrase. It is certainly legitimate and customary to say "the dip of the line of intersection of the vein with the cross-vein," "the dip of the incline shaft" (which is often different from the dip of the vein), or "the dip of the line of intersection of the vein with a vertical plane having a course—" (any other plane than the one showing the maximum dip).

* Received Oct. 13, 1908.

The axis of the ore-body has a different direction from the vein as well as a different dip; yet we do not think of giving this new azimuth a different name. It is still an azimuth, an angle measured in a horizontal plane from the meridian, and the dip of the ore-body is the angle measured in the vertical plane from the horizontal.

To describe the situation completely and exactly (as, for example, it might be necessary to do in a case in court), we must give both the course and dip of the ore-body as well as the strike and dip of the vein. Now, introducing the term "pitch" as the angle measured in the plane of the vein, and taking the same example and angles heretofore used in this discussion (pp. 326 and 327), but making the dip west to correspond with the figure on p. 899, the complete definite statement of the case would be: "The vein has a strike due north (*AD* and *BC*) with a dip 45° west (angle *ABE*). The ore-body has a pitch 45° north (northerly) in the vein (angle *CAD*). This gives the axis of the ore-body a course N. 35° 16′ W. (*EC*) with a dip of 30° (*ACE*)."

The point I would maintain is that the angle in the plane of the vein is the only function which calls for a different name, and unless "pitch" is used for this angle there is no legitimate use for the term.

2. Considering the practical phase of the case:—No doubt the term "pitch" has been much used without a thought as to which of the two definitions would be applied. For veins having a nearly vertical dip, the difference is small, and would not be apparent in ordinary scaling from a map. With such veins the same thing is experienced in the common use of the vertical projection and the projection in the plane of the vein. The mine-surveyor makes a projection of the vein-workings on a vertical plane. This projection must be made in any event, if only as a step in the process of making a projection in the plane of the vein. If the vein-dip is not far from the vertical, no projection in the plane of the vein is ever made; the engineer uses the other projection for what it really is, while the miner uses it as if it were a projection in the plane of the vein; and the practical purpose of both is served. The miner takes measurements in stopes, raises, and winzes, and scales them on the map as though they were vertical distances; and the error is too

slight to be readily apparent on the scale of the map. In this manner a mine-map is often posted up in pencil, and answers for temporary use until the engineer goes over the work and makes it correct. Just where the limit of practical accuracy is reached in such a procedure depends, not only on the dip of the vein but on the scale adopted, and the purpose for which the sections are used. As the dip approaches 45° a projection in the plane of the vein becomes a practical necessity for the use of the foreman of the mine, or before a jury, even though the engineer may never use it in his own computations. I think it undeniable that mine-managers, superintendents, and all who look daily at both the underground workings and the maps, will do their thinking as well as their measuring mostly in the plane of the vein. The miner will consider that his raise or winze has so far to go in the vein to reach the next level, and will measure the area of vein extracted. In short, he measures, plans, and estimates in the plane of the vein; and our mathematical engineering practice can never change this habit.

Moreover, most strictly practical men would probably prefer to consider the pitch-angle from the vertical, instead of from the horizontal, just as they prefer to use the "hade" angle where the dip is greater than 45°. In both cases they largely employ the co-ordinates, rather than degrees, in describing an angle. But since custom has fortunately simplified matters by discarding the use of the "hade" angle in technical literature, the pitch should correspondingly be measured from a horizontal line.

As Professor Louis says, an additional calculation is required to determine the angle in the plane of the vein, since the horizontal and the vertical angles are the first to be determined by survey; but since it is seldom possible, in any extensive mine-workings, to measure directly either strike, dip, or pitch, and all are arrived at by a series of trigonometrical operations following a survey, the addition of one more step in the problem seems of little consequence. Besides, the "pitch," whichever definition is taken, is usually scaled from the map with a protractor—a method which answers the purpose, in most cases, without exact figures.

Professor Louis, on p. 902, advocates the practice of using the horizontal and vertical functions in measuring ore-bodies—

"horizontal thickness into vertical height into length along strike." Thus, he would construct a projection on the vertical plane *ADFE*, Fig. 2, and the vertical angle to the axis of the ore-body projected on this plane is *DAF*, which is different from either his definition of "pitch" or that of Dr. Raymond. In our example the "pitch" angle is, according to Professor Louis, *ACE* (30°); according to Dr. Raymond and myself, *CAD* (45°); and according to this last conception it is *DAF* (35° 16′). I have talked with one engineer of large experience who claims the last as his understanding and use of the term. I agree with Professor Louis as to the advantage of using the horizontal and vertical functions in engineering practice for

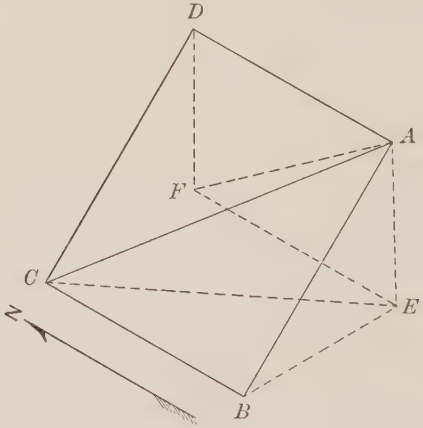

FIG. 2.—THE SAME AS FIG. 1, WITH THE ADDITION OF THE VERTICAL PLANE *ADFE*.

determining ore-quantities and values; my experience has drawn me more and more in that direction; but I fail to see that this practice has any bearing on the question at issue, either technically or practically, unless it is to support the last view above mentioned, which contradicts both of us.

I cannot see that there is any practical advantage in taking the vertical angle of the axis of the ore-body as "pitch," because a shaft is occasionally to be constructed along this line. Such a case is rather an exception; against one mine so operated there are, according to my experience, at least fifty which are opened either by inclines following the vein on its dip, with drifts along the vein, or by vertical shafts with cross-cuts.

If the greater number of ore-bodies were "chimneys" or

"pipes," Professor Louis's definition might be the more practical; since, in that case, the stope-sheets would be drawn on a vertical plane having the course *CE;* but certainly such ore-bodies are rare as compared with ordinary lenticular deposits in veins.

Harold N. Lawrie, White Salmon, Wash. (abstract of communication*):—The questions seem to be: (1) Which of the two definitions of "pitch" is theoretically the better—that is, which would be preferable if the term were a new one, and we were entirely free to fix its meaning? and (2) which of them is now employed by the larger number of intelligent English and American mining engineers? The second, however, in a certain sense supersedes the first, demanding prior investigation. For it is, as Dr. Raymond suggests, much more reasonable to attach a new name to a new conception than to limit or change the meaning of an old and familiar term.

I feel sure that, however engineers may actually determine a pitch for their own maps and calculations, the majority of them are obliged to use the term in the sense given by Dr. Raymond, whenever they talk with the foremen or shift-bosses or working miners; for unquestionably the men who are in contact with the underground work refer their courses and distances to the plane of the bed or vein in which it is carried on. Not only is it wise to accept from them that practical and natural meaning, but it will also prove convenient and useful. I have found it necessary in describing ore-shoots and lenses. When a vein has repeatedly changed its course, and the fracturing attending these changes has led to the formation of ore-bodies, the direction and inclination of such bodies as measured in the plane of the vein is an essential descriptive element, with an important theoretical meaning besides. For it directly indicates the relation of the bodies to one another, and may thus aid us in determining their probable common source, and the direction and distance at which they may be successively encountered in mining.

I agree with Professor Louis that projection on horizontal and vertical planes is a most accurate and rapid method for the calculation of reserves. But, if used exclusively, it tends to hinder the imagination in forming a picture of the deposit and

* Received Aug. 27, 1908.

the mine, and their relations to neighboring formations. For this purpose, it is well to retain the practice of referring to the plane of the bed or vein, and thinking about the deposit or the mine in terms based upon that plane. Moreover, contract-work not on a level is usually set by the linear foot on the dip, and any other conception would be unsatisfactory to contractors, inasmuch as every change in dip would occasion a new estimate. On the whole, it seems to me that engineers will have to use both systems, but not apply "pitch" to both.

I have had occasion to use Professor Louis's angle *ACE* as a necessary term in the description of a geological occurrence; and I was embarrassed by the lack of a name which would distinguish it clearly from the accepted meaning of "pitch." I hastily coined the somewhat clumsy phrase, "the dip of the pitch"—protecting myself at the same time by adding a diagram. A name for this angle *ACE* would be most desirable; for, as Professor Louis says, it is an important element in the development of certain classes of deposits. Yet I must confess that I can recall but few instances of that kind in this country or abroad, as compared with the numerous instances in which mines are laid out without reference to the angle *ACE*.

Dr. Raymond:—Having introduced at several points, for the convenience of the reader, my remarks upon the individual contributions comprised in the foregoing discussion, I shall say little in the way of conclusion. Indeed, I do not see why this interesting comparison of views should be regarded at this time as requiring any conclusion. I should welcome further additions to it.

One such addition I venture to make here, by citing the following paragraph from the recent book of T. A. Rickard[1]:

"The words *dip*, *hade*, and *pitch* are used confusedly. It may be well to apply *dip* to the inclination (from the horizontal) of strata, veins, and faults, rejecting *hade* as a term no longer necessary and only likely to make confusion, because it refers to the angle from the vertical. The angle made by an ore-shoot in the plane of the vein is its *pitch;* this also should be measured from the horizontal."

Mr. Rickard's wide and long experience as a mining engineer, and as the editor of technical journals, gives weight to his opinion; and I regard it as the more valuable on this par-

[1] *A Guide to Technical Writing*, San Francisco, Cal., p. 46 (1908).

ticular point, because it was expressed independently of the present discussion.

Without assuming to close the debate at this stage, I think it fair to say that the following facts are so far indicated:

1. That the interchangeable use of "dip" and "pitch" still survives, in practice if not in literature.

2. That, where a distinction is made between them, the general practice of engineers, as well as miners, in this country is to measure the angle of the pitch on the plane of the dip.

H. L. Smyth, Cambridge, Mass. (communication*):—I have been much interested in the recent discussion of dip and pitch. My own impression agrees with that of Dr. Raymond—namely, that the term "pitch," which is certainly in nearly if not quite universal use among engineers in America, is commonly employed rather loosely to express in a general way the fact that an ore-body in a vein or other tabular locus does not usually continue down on the line of the dip, but departs from that line to one side or the other. This departure is usually stated as so many feet measured horizontally along the vein in so many feet of descent, rather than as an angle, but when it is spoken of in degrees the angle meant is not the angle measured from the line of dip, but the complementary angle measured for greater convenience from the horizontal. Also in regard to the plane in which the angle designated as the angle of pitch is usually measured, my experience is the same as Dr. Raymond's, and indeed for 15 years I have been following that usage with my students. But it is undeniable that the word is also used in another sense, and that it is not an uncommon practice to apply it to the corresponding angle measured in the planes of projection, especially the longitudinal vertical projection. That is to say, the angle often designated as the angle of pitch is not merely the angle measured from the horizontal in the plane of the vein, but the vertical projection of that angle as well. It is true, as Dr. Raymond says, that one occasionally finds the openings of a metal-mine shown on the plane of the vein, as, for example, at some of the Lake Superior copper-mines. In such cases the term "pitch" is naturally applied only to the angle measured in the plane of the lode.

* Received Jan. 14, 1909.

Is there not something to be said in favor of keeping this term pitch of a comfortable looseness, so that it may continue to include this extension of meaning which experience has found to be desirable? It ought not to be forgotten that neither the obliquity of the ore-body to the dip nor the inclination of its axis to the horizontal plane is directly observable or measurable. We discover that the ore-body is oblique as it is gradually developed and mined, and the fact and the amount of obliquity are brought out with clearness only when the underground openings are mapped. The angle of pitch, as defined either by Professor Louis or by Dr. Raymond, may be determined by computation or graphically by construction, but usually only after this platting is completed. The representation of the boundaries of the ore-body on the maps is therefore an almost indispensable preliminary to their determination. Furthermore, it is only very rarely that it is necessary to determine them at all. In ninety-nine cases out of a hundred the chief reason for determining the obliquity of an ore-body is to aid us to plan ahead, to lay out development-work on lower levels, to recognize clearly the probable relation of the ore-body to property-lines in depth, and in general to make intelligent forecasts. Now in order to do this it is not necessary to know the angle of obliquity (or its complement) measured in the plane of the vein, and still less the angle of plunge of the axis measured in the vertical plane. Ordinarily we merely prolong on the map the limits of the ore-body from the upper levels into the unknown territory below, without measuring any angle whatever. When it becomes desirable to do so, the only angle that need be measured is the angle before us on the map itself. Even if there were any good independent way of determining the angle of pitch, according to Professor Louis or according to Dr. Raymond, without having this angle on the map, either of those angles would be unavailable for ordinary purposes until it had been projected again into the angle on the map. In short (except in the case mentioned by Professor Louis, which can arise only very rarely), the angle on the map is the only angle that we need to know and name. Why, then, should we restrict the name "pitch," which in current use now includes this angle, to angles which hardly ever need be known or talked about? In fact, if we could now restrict the

term to either of the angles suggested, would it not in time naturally and inevitably be extended to include its projection? And is it not better, if Professor Louis's angle needs a name, to give it a new one, such as "plunge?"

Professor Louis's proposal seems to me objectionable for still further reasons. In the first place, while I have not seen Professor Louis's original paper, it seems to be clear, from the quotation given by Dr. Raymond, that, at the outset, the object he had in mind was to propose a term to describe the obliquity between the line of maximum dip and the direction of an ore-shoot:

> "It is well known that the ore-shoots in a mineral vein do not, as a rule, coincide with the dip of the vein, but make an angle with it; in the same way, lenticular masses often are oblique to the direction of maximum dip of the plane in which they lie. We have no word to designate this obliquity in either case."

Now the angle which, as a matter of fact, he proposes to designate by the term "pitch" has no relation whatever to the "obliquity," which he recognizes as the thing for which a name is desirable, and only a somewhat remote relation to the dip.

Again, Professor Louis defines the angle of pitch as the angle made by the axis of the ore-body with the horizontal plane. This seems to me to be objectionable, for the reason that the axis of an ore-body is a line which has no physical or objective existence, and therefore cannot be recognized underground. Indeed, in most cases, the only way in which the axis of an ore-body can be determined is to draw it on the maps after the boundaries of the deposit have been outlined. It is fixed, therefore, somewhat arbitrarily, and we may doubt whether in practice there are many ore-bodies, the axes of which any two men would fix precisely alike. On the other hand, the boundaries of an ore-body have an actual physical existence. They are definitely determined by drifting, in the ordinary course of opening a level, and by the termination of the stopes between the levels. Frequently, therefore, the pitch of an ore-body is more easily as well as more accurately designated by the inclination of its boundaries than by the inclination of its axis. Of course, these boundaries are by no means always parallel; indeed, in an ore-body which becomes

either longer or shorter with depth, they are necessarily not parallel. In such cases, I think it is common to speak of one boundary of an ore-body as having one pitch, and of the other as having a different pitch. The definition, therefore, should be sufficiently broad to include the inclination of the boundaries as well as that of the axis.

I confess that I do not understand what Professor Louis means, when he says (p. 902) that the use of the term pitch, which he advocates, "is more scientific, because when we speak of the angle of inclination of any straight line to the horizontal, we always mean the angle as measured in a vertical plane." The two angles, to one of which it is proposed to apply exclusively the term "pitch," are not the same angle which the opposing parties would measure in two different ways. One of them is the angle made by the axis of the ore-body with the horizontal plane, and is necessarily measured in a vertical plane. The other is the angle between the axis or boundaries of the ore-body and a particular horizontal line drawn in the plane of the vein, and is necessarily measured in the plane of the vein. Therefore, if Professor Louis's statement means anything, it must mean that an angle lying in a vertical plane is in some way more deserving of scientific consideration than one lying in a plane not vertical—a proposition which, of course, he could hardly mean to maintain.

A New Theory of the Genesis of Brown Hematite-Ores; and a New Source of Sulphur Supply.

Discussion of the paper of H. M. Chance, p. 522.

Charles Catlett, Staunton, Va. (communication to the Secretary*):—Mr. Chance's suggestions that the brown hematite-ores of the Potsdam formation are due to the alteration in place of iron sulphide, and that their extension may reasonably be expected to contain valuable deposits of unaltered sulphide, are interesting; but the facts marshaled by him are too limited

* Received Oct. 19, 1908.

to justify more than the hope that they may serve as a commercial source of sulphur.

It is true that these deposits have been the source of large amounts of iron-ore; and it is common, as stated by him, for the mining operations to extend to a shallow depth only; but this is largely due to the fact that the surface-material represents an accumulation from the breaking down and collection of the ore which formerly reached a much higher level. With increasing depth comes a decreasing thickness of valuable material. This is especially appreciated by a glance at Mr. Chance's drawings. This is the reason why so many disappointments have resulted when these ores have been followed to greater depths.

Even if the ore is largely derived from oxidation in place, it is impossible to ignore the great number of concretionary forms, as well as those showing the evidence of secondary solution and deposition now taking place, such, for instance, as the part-replacements of fragments of quartzite derived from the underlying beds. Concretionary form does not necessarily mean long transportation, and may be a variation of material in place; but commonly it is an evidence of collection, and tends to imply that more or less concentration has taken place. This fact would also argue against an equally extensive development in the unaltered rocks. These two questions raise a very serious doubt whether, even if these deposits can be traced to sulphide, they will prove of commercial value.

A more serious doubt is whether the iron which forms the valuable constituent of the ore was originally in the form of sulphide. That it is possible for the iron in these sedimentary rocks to have been deposited as sulphide is true, though a brief glance at the literature of the subject does not disclose common instances of the occurrence of workable deposits of iron sulphide which there is reason to believe have been so formed. The common form in which we might expect the iron to be deposited would be the carbonate; and there are at least some instances in which the *prima facie* evidence indicates that the ores of this formation were so deposited. Mr. Chance says that both the sulphide and the carbonate have been encountered in these ores, and (if there has been no typographical error) that the typical mine at Easton, Pa., contains very pure car-

bonate mixed with limonite in the lower workings, while "the presence of pyrite has not been noted." He explains that the carbonate was derived by alteration from the sulphide. It would be a valuable support to this contention if it can be shown that it is common to find masses of carbonate in brown-ore deposits, which are known to be the outcrop of relatively pure iron sulphide.

In prospecting the ores near Vesuvius, Rockbridge county, Va., the slates overlying the Potsdam quartzite were shown to be highly calcareous, and the ore, which was lying practically on the quartzite, was traced until it ended in an impure carbonate. No sulphide was noted.

At the Sugar Hill mines, near Cartersville, Ga., both sulphide and carbonate are encountered in the lower workings of the open-cuts; but the carbonate is enormously greater in quantity. It is interesting to note that at one time any carbonate encountered was thrown away under the belief that it was a siliceous impurity. The fine tailings from the old washers were examined and an effort at concentration was made, which resulted in showing that most of the light-colored material which was supposed to be silica was iron carbonate.

A large mass of soft white carbonate carrying minute but perfectly formed crystals of pyrite, and gradually altering to a brown hematite on the exterior, was noticed. On analysis, it gave the following result:

	Per Cent.
Metallic iron,	36.63
Silica,	1.26
Alumina,	0.21
Phosphorus,	1.95
Sulphur,	0.706
Calcium oxide,	8.45
Manganese,	2.64

If the sulphur is calculated as pyrite, the phosphorus as calcium phosphate, and the manganese, together with the balance of lime and iron, as carbonate, the footing approximates 100 per cent.

Near water-level, at one place, the extension of the ore as at present worked is a heavy bed of carbonate.

These facts indicate that in these localities the original formation of the iron was as a carbonate.

Probably the strongest theoretical reasons for believing that the predominating form in which the iron was deposited was carbonate, are those given by Penrose[1] in discussing the origin of manganese in this formation. He points out that manganese is commonly deposited as carbonate, and rarely as a sulphide; and that iron and manganese carbonates are isomorphous and have a strong tendency to combine into a homogeneous mass and to crystallize together in varying proportions. This is certainly the most natural way to account for the common occurrence at this horizon of ore consisting of homogeneous masses of iron and manganese oxides in every possible proportion.

Mr. Chance does not give any specific reasons for his belief that the Oriskany ores are also gossan, or the oxidized outcrop of beds of sulphide. It would be interesting to know whether he has ever encountered any undecomposed sulphide in the Oriskany ore. I have never had my attention called to any such occurrence, but I have had occasion to notice other facts which tend to indicate a different source for this ore.

The Devonian slates and shales in the section of Virginia carrying the Oriskany ore in its best state of development were originally of great thickness, as is shown by the sections which have not been eroded. These slates, at many places, are heavily charged with iron sulphide. The waters which flow from them also carry large quantities of iron either as a sulphate or as a carbonate. (In the mines of the Glen Wilton furnace, the water coming from the slates into the mine-workings is heavily charged with iron sulphate.) Lying beneath the slates is the Oriskany sandstone, which in that section is commonly porous, persistent, and relatively thin. Beneath the Oriskany sandstone is a series of limestones carrying interbedded chert. There are local variations, but this is the normal occurrence. These strata are folded and bent in all directions, but are commonly tilted at a high angle and lie on the slopes of the subordinate ridges in their relative geological order, the limestone being beneath. The waters, which are heavily charged with iron sulphate from the decomposition of the slates, find a ready channel in the sandstone, where they mingle with the waters derived from the limestone, and in

[1] *Report of the Arkansas Geological Survey for* 1890, vol. i., p. 569 *et seq.* (1891).

turn aid in the decomposition of the latter. The result would be the formation of iron carbonate, and then of iron oxide and calcium sulphate, with the disintegration of the limestone. The calcium sulphate being readily soluble, would usually be entirely removed. The most common plane of deposition would be the contact between the limestone and the sandstone, leaving a part of the sandstone as a ferruginous hanging-wall.

The chain of facts in support of this theory is: the abundance of iron sulphide in the slates; the fact that the waters from these slates are heavily charged with iron sulphate; that these waters do deposit iron in contact with the limestone; the part or complete disappearance of the limestone, with the retention of its cherty framework as a foot-wall to the ore; and the occurrence of crystals of calcium sulphate in the lower levels of the mines.

An interesting instance in this connection is seen at the quarry of the Low Moor Iron Co., which is operating on these limestones. The strata at the quarry are nearly horizontal, and while the Oriskany sandstone itself carries no ore, the waters passing through the sandstone precipitate a gouge of soft good ore directly on the limestone. At this point the occurrence of stalactites of brown hematite, several feet in length and attached to the sandstone, has been noticed. Another fact of interest is that while I have prospected these measures extensively, and though overturns in this section are exceedingly common, I have never found workable ore in the Oriskany sandstones where the slate was underneath.

The value of the Oriskany ore-deposits at any point must have been determined by the distribution of the sulphide in the slates, and the opportunity for oxidation and solution; the extent and character of the sandstone acting as a favorable conduit; the extent and character of the limestone acting as a precipitant; and the effect of elevation and distortion as determining the lines of oxidation, drainage, precipitation, and concentration. The result is, that while the Oriskany ores are found over a large territory at a particular geological horizon, they vary greatly in their development, and through only a small percentage of the extent of that horizon are they found in workable condition.

An Unusual Blast-Furnace Product; and Nickel in Some Virginia Iron-Ores.

Discussion of the paper of Frank Firmstone, p. 547.

JOHN J. PORTER, Cincinnati, Ohio (communication to the Secretary *):—The remarks of Mr. Firmstone concerning the presence of nickel in Oriskany ores, call to my mind some additional evidence on this point.

In 1903, while engaged in the determination of zinc in a number of samples of ore from the Oriskany mine of the Alleghany Ore & Iron Co., I was surprised to discover the presence of nickel and cobalt. As far as I was then able to discover, their presence had up to this time remained unnoticed. The amounts ranged from 0.05 up to 0.10 per cent. of the combined oxides, and while no attempt was made to separate the metals, the presence of both nickel and cobalt was conclusively proven.

Since that time I have learned that traces of these metals are of quite general occurrence in Virginia iron-ores, and more especially in the ores of the Potsdam formation from the western slopes of the Blue Ridge. In certain manganese-ores from this formation, I have found as high as 0.50 per cent. of nickel and cobalt oxides.

Any one interested in this subject will find further information in a volume [1] written by Andrew S. McCreath, and printed in 1884 for distribution by the Norfolk & Western railway. A number of complete and accurate analyses of Virginia iron- and manganese-ores are given, and practically all show nickel and cobalt in appreciable quantity.

Still another source of information on the occurrence of nickel in iron-ores in the United States, for which I am indebted to Charles Catlett, is found in the Tenth Census. Several analyses of Oriskany ores are given,[2] all of which show the presence of nickel, with less cobalt, while the single representative from the Potsdam formation shows nickel, with no cobalt.

* Received Oct. 8, 1908.

[1] *The Mineral Wealth of Virginia.*

[2] Report on the Mining Industries of the United States (Exclusive of the Precious Metals), etc., vol. xv., p. 601 (1880).

Experience with the Gayley Dry Blast at the Warwick Furnaces, Pottstown, Pa.

Discussion of the paper of Edward B. Cook, p. 705.

Edgar S. Cook, Pottstown, Pa.:—Many friends and acquaintances seem to be under the impression that the Warwick Iron & Steel Co. received a license from Mr. Gayley, free of cost, as an inducement to adopt the process. This impression is not correct, since the license was paid for, and I think it was the first payment that Mr. Gayley received for the use of his process.

As president of the Warwick Iron & Steel Co., I am responsible for its earnings, and with only the experience at the Isabella furnaces as a guide, the directors of the company considered that there was a great risk in expending the large sum of money required for the plant. They, however, acted favorably upon my recommendation, and to this extent I am responsible for the installation at the Warwick furnaces. I made the best agreement possible with Mr. Gayley, and naturally our company received certain advantages. It will be contrary to all experience if subsequent plants do not show some improvements and economies in construction. It is to our indirect benefit that all subsequent licenses granted by Mr. Gayley should be charged at a much higher rate than we paid; and I trust that he will see to it that all such licenses granted, especially to companies making merchant iron, shall be at a maximum figure.

The Warwick Iron & Steel Co. is not interested in any way in exploiting the merits of the Gayley dry-blast process; on the contrary, looking at the subject from a narrow and selfish standpoint, it is to the advantage of our company that no merchant blast-furnace competing with our iron-product should be equipped with a dry-blast plant.

There is a clause in the contract with Mr. Gayley, whereby our company agrees to furnish him with all the data derived from the operation of the dry-blast plant, and that he or his

representatives shall have access to our records. I therefore only carried out our contract with Mr. Gayley in consenting to have my son, Edward B. Cook, prepare the paper just read.

As a matter of history, I have taken an active interest in the use of dry blast for many years. In 1889 I commenced to ascertain regularly the quantity of moisture in the atmosphere. Tests were made at intervals of every two or three hours, day and night, and daily records were gathered together, first into weekly and then into monthly reports. After a while I was able to utilize the daily observations in managing the furnace with better results, as to both fuel-economy and uniformity of product.

At that time I was not aware that the desirability of removing moisture from the atmosphere for use in blast-furnaces had been referred to in an English publication of 1800, or that it had been the leading subject of the first address of the first President of the Iron and Steel Institute of England. I had overlooked the fact that the subject had been discussed in Bell's *Chemical Phenomena of Blast-Furnaces*, and summarily dismissed as being chimerical from a practical and economical standpoint.

"Grains of moisture per cubic foot of air blown into furnace per minute" did not convey to me any definite impression, but when I found from a study of my data that many pounds of water were entering the tuyeres for every ton of iron made, the subject assumed a serious aspect. John Birkinbine became associated with me in helping to solve the problem of removing the atmospheric moisture by chemical means, but the conclusion which we deduced was that the expenditure for constructing and operating the plant would scarcely be justified by the pounds of coke saved by abstracting the moisture from the air.

No one had given sufficient consideration to the value, measured in pounds of fuel per ton of iron, of uniformity in the moisture-content of the air blown into the furnace as affecting regularity of working. I doubt whether Mr. Gayley at that time had looked beyond the mere saving of fuel required for converting the moisture into oxygen and hydrogen at the expense of the temperature of the crucible.

During the course of Mr. Gayley's experiments, I kept in

more or less close touch with them, and frequently expressed my doubts that he would ever accomplish his object. He, however, had the faith, together with the energy and perseverance, to accomplish what had been talked about for a hundred years, and to demonstrate the means of removing the moisture in a practical way.

The Warwick Iron & Steel Co. is now entirely satisfied that the investment for the installation and operation of the Gayley dry-blast plant has not been in vain.

The Uniform Nomenclature of Iron and Steel.

A Discussion of the Report of Committee 24 of the International Association for Testing Materials, presented at the Brussels Congress, 1906, and republished in *Bi-Monthly Bulletin*, No. 20, March, 1908, pp. 227 to 237, but not included in this volume.

Henry D. Hibbard, Plainfield, N. J. (communication to the Secretary*):—The definition of steel, given on p. 235,

[Steel, iron which is malleable at least in some one range of temperature, and in addition is either (*A*) cast into an initially malleable mass; or (*B*) is capable of hardening greatly by sudden cooling; or (*C*) is both so cast and so capable of hardening. Variety *A* includes also molten iron which, if cast, would be malleable, as do its two sub-varieties, "ingot-iron" and "ingot-steel." (Tungsten-steel is malleable only when red-hot.)]

does not seem to fit perfectly the requirement that it shall include all known varieties of steel and exclude everything else. The first requirement, that it shall be malleable at some temperature, is also possessed by some varieties of gun-iron—that is, mottled cast-iron, of which cannons have been made in the past by the South Boston Iron Co. and others. Gun-iron also meets the requirement (*A*) in being cast into an initially-malleable mass. The malleability was indeed not great, nor was it made use of except to demonstrate the quality of the iron, but it approached in measurable degree that of some of the alloy-steels. As for requirement (*B*), some puddled-irons having from 0.2 to 0.3 per cent. of carbon may be hardened greatly, although this property is not utilized.

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* Received Mar. 3, 1908.

Of course, unusual properties in some of a class of irons which give them no value and do not add to their usefulness are of no practical importance or effect except as they interfere with the definition.

It has seemed to me that the usual meaning conveyed in using the word steel, rests on its unique valuable properties which are actually utilized, and that the following definition meets fairly well the case in covering the commercial steels and perhaps all steels as we actually know them.

"Steel is a variety of iron which, at some temperature, is malleable to a useful degree; it is made by a fusion-process, or when not so made is capable of being hardened to a useful degree." By the term "useful" in this definition is meant such as is actually utilized in practice.

The theoretical objection to this definition is that there is no dividing-line between useful and non-useful degrees of malleability and hardening, but it is not at present difficult to say that a given piece of iron the properties of which are known is within or without the limits of the definition, and if doubtful cases arise in the future some way may then be found to define them.

William Kent, Syracuse, N. Y. (communication to the Secretary*):—I suggest that the terms "weld-iron," "ingot-iron," "weld-steel," and "ingot-steel" be starred as obsolete or obsolescent. They were arbitrarily introduced by the International Committee at Philadelphia in 1876, and were adopted by a few writers of books and technical papers at that time, but they have never been introduced in commerce.

I think it is useless to make any distinction between the generic and the specific definitions of cast-iron. Cast-iron is iron containing so much carbon or its equivalent that it is not malleable at any temperature, whether it is in the shape of melted metal in the ladle, or is run into pigs, or is in the form of castings. The long paragraph attempting to explain the difference between the generic and the specific use of the term "cast-iron" might be applied to any other material whatever, such as stone, brick or wood. Thus, generically, stone is a hard material found in the earth's crust; and specifically, stone might be de-

* Received Apr. 7, 1908.

fined as the same material made into the form of sculptured blocks. The term "cast-iron" is used in common language to mean a metal which is distinguished from steel and from wrought-iron, and whether it is a specific or a generic term makes no difference.

I regret to see that the Committee recommends that a line be drawn between cast-iron and steel at 2.20 per cent. of carbon. The present Committee should have profited by the experience of the former International Committee (Philadelphia, 1876), and realized that no committee can change the meaning of words commonly used in the English language. That Committee failed in its attempt to get people to distinguish between "ingot-iron" and "ingot-steel," and the present Committee will fail to establish 2.20 per cent. of carbon as the dividing-line between cast-iron and steel.

Professor Wedding's dissent from drawing any line between cast-iron and steel is exactly right, if he means that a line should not be drawn by a committee. The definition of terms will be settled by commerce and by common usage and not by a committee. In the United States, as in Germany, every metallic product of the blast-furnace is called "pig-iron" or "cast-iron," except possibly when the product has a specific name, such as "spiegeleisen."

With regard to the statement that cast-steel is the same as "crucible-steel," the term "cast-steel" is gradually being expanded by commerce to have the same significance as the adjective "steel-cast." A cast-steel crank would be a crank cast out of either crucible, open-hearth, or Bessemer steel, distinguishing it from a forged steel crank, and the adjective "steel-cast" would not be applied to it by one person out of a hundred thousand. The statement that the term "cast-steel" should be avoided because "confusing and because a temptation to fraud" had better be left out. Manufacturers of steel sell their product for what it actually is under its proper name, "crucible," "open-hearth," or "Bessemer," and as to products made out of forged steel, such as cutlery and springs, there is no reason for there being a temptation to fraud, because the hardware trade does not distinguish between these grades of steel in selling these products.

"Plate-iron. The name applied in Great Britain to refined

TABLE I.—*Classification of Iron and Steel.*

Generic Term.	Iron.				
How Obtained.	Cast, Or obtained from a fluid mass.			Wrought, Or welded from a pasty mass.	
Distinguishing Quality.	Non-malleable.	Malleable.		Will Not Harden.	Will Harden.
Species.	Cast-Iron.		Cast-Steel.	(7) Wrought-Iron.	(8) Wrought-Steel.†
Varieties.	(1) Ordinary castings.	(2) Malleable cast-iron, obtained from No. 1 by annealing in oxides.	(3) Crucible, (4) Bessemer, and (5) Open-hearth steels. (6) Mitis.*	*a.* Obtained by direct process from ores, as Catalan, Chenot, and other process irons. *b.* Obtained by indirect process from cast-iron, as finery-hearth and puddled irons.	Obtained by direct or indirect process, as German, shear, blister, and puddled steels.

* No. 6. Mitis is the name given to a new product (having the same general properties and produced by the same processes as soft cast-steels) made by adding an alloy of aluminum to melted wrought-iron or soft steel before pouring.

† No. 8. Wrought-steel is almost an obsolete product, having been replaced in commerce by cast-steel.

Sub-varieties of Nos. 3, 4, and 5, soft, mild, medium, and hard steels, according to percentage of carbon, the divisions between them not being well defined.

Cast-iron usually contains over 3 per cent. of carbon; cast-steel anywhere from 0.06 per cent. to 1.50 per cent., according to the purpose for which it is used; wrought-iron from 0.02 per cent. to 0.10 per cent. The quality of hardening and tempering which formerly distinguished steel from wrought-iron is now no longer the dividing-line between them, since soft steels are now produced which, by the ordinary blacksmith's tests, will not harden. All products of the crucible, Bessemer, and open-hearth processes are now commercially known as steel.

cast-iron." I think the name is not quite obsolete in this country, for cast-iron is still for some purposes refined in the run-out fire and run into plates which are called "plate-iron."

"Refined iron" and "double refined iron" are terms which might be inserted in the definitions, meaning wrought or puddled iron which has been refined by one or more re-rollings.

The question of establishing a boundary between steel and iron was thoroughly discussed by the Committee of 1876, which decided that the line should be between hardening and not-hardening. This decision was never accepted in commerce, and it is just as useless now to try to draw a line at 0.32 or 0.20 per cent., or any other percentage of carbon. Mr. Pourcel is entirely right in saying that steel should include all forms of iron freed from slag by fusion and cast in a malleable condition. This is exactly the definition for which Holley contended

more than 30 years ago. It is the definition that has been consistently adhered to in the trade ever since Bessemer steel was first made, and it has been incorporated in the United States tariff laws. No matter what any committees may determine as the boundary-line between steel and iron, commerce and the usage of the people will determine finally the definition.

Table I. is a classification of iron and steel which is substantially the same as the one I prepared a few years ago.[1] I think it is still a good classification and I have not been able to improve it.

Prof. Henry M. Howe, New York, N. Y. (communication to the Secretary *):—Mr. Hibbard's definition of steel would be the best yet offered were it not for its one defect, the vagueness or equivocalness of "fusion-process." Puddling is surely in one sense a fusion-process, and so are the charcoal-hearth or knobbling-processes. But what he says suggests the following definition, which seems to me a good one:

"Steel: any variety of iron which is usefully malleable at least at some one temperature, and is either cast when molten into a malleable mass, or if not so cast can be much hardened by sudden cooling." It is not necessary in a definition that quantitative bounds should be set up, for instance, that the exact height at which dwarf apple-trees cease and standards begin, or at which hills cease and mountains begin, should be stated. These quantitative measures are proper for specifications and contracts, but are not necessary for purposes such as the present.

Professor Kent's criticisms seem in general to be based on misinformation.

He is mistaken in supposing that the term "ingot-iron" has not been introduced in commerce. It is still used in commerce and may play an important part.

He is mistaken in supposing that it is useless to make a definition between the generic and specific meanings of cast-iron. This error of his is evidently based on his overlooking the fact that a large body of men, including some distinguished writers, maintain that pig-iron is not cast-iron, and that "cast-iron" can

[1] *Railroad and Engineering Journal*, vol. lxi., No. 4, p. 158 (April, 1887).

* Received May 8, 1908.

be used only to designate pig-iron which has been remelted and cast into castings. In short, there is an important group of men which recognizes at present only the specific meaning. On the other hand, there is a very large group which recognizes the generic meaning. Both meanings are in existence, as the committee knows, and its clear duty is to co-ordinate these two meanings.

Professor Kent is in error in supposing that the committee does not realize "that no committee can change the meaning of words commonly used in the English language." Our committee distinctly recognizes that, and has made no attempt to change the meaning of any word. Professor Kent, of course, understands that to propose a line where none previously existed is not to change the meaning of words. If the committee had attempted to change a line previously existing then it would have deserved Professor Kent's censure, but it has attempted nothing of the kind.

Professor Kent knows that "the present committee will fail to establish 2.20 per cent. of carbon as the dividing-line between cast-iron and steel." How does he know that?

Professor Kent is mistaken in supposing that, if the usage of the public must dispose, a committee cannot profitably propose.

Professor Kent is mistaken in supposing that wrought-steel is an almost obsolete product. An enormous quantity of blister-steel is bought and sold currently, and this will probably continue to be the case. This is certainly wrought-steel.

Henry D. Hibbard, Plainfield, N. J. (communication to the Secretary *):—Professor Howe's criticisms of my definition of steel are not, it seems to me, well founded, for the following reasons: The term "fusion-process" has been used during the past quarter- or half-century to distinguish the crucible, Bessemer and Siemens-Martin processes from the puddling- and cementation-processes, and probably no one at all conversant with the art of iron-metallurgy would now mistake its meaning.

Moreover, no steel-metallurgist is in doubt of the meaning of the term "hardened" as applied to steel. If it were really necessary to describe what is meant by the term then it would not be enough to say "hardened by sudden cooling" as Pro-

* Received Aug. 3, 1908.

fessor Howe proposes. Plunging a piece of steel at ordinary temperature into liquid air will subject it to sudden cooling but without hardening in the sense intended. One would have to say something like this: hardened by sudden cooling (rate of cooling unknown) from a temperature at least as high as a certain critical temperature, which depends on the composition of the variety of steel in question, etc.

Again, if the limitation of usefulness be proper as applied to malleability, why is it not also proper as applied to the hardening capacity of puddled and cemented steels?

I think it may also be maintained that the definition, to be of value, should be exact.

Prof. Henry M. Howe, New York, N. Y. (communication to the Secretary*):—Mr. Hibbard's defense of the term "fusion-process" on the ground that "no one at all conversant with the art of iron-metallurgy would now mistake its meaning" seems to me inadequate. It is not enough that a definition should be comprehensible to experts. It should be clear and unmistakable to educated people in general, including those who are not experts.

* Received Mar. 4, 1909.

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[NOTE.—In this Index the names of authors of papers are printed in small capitals, and the titles of papers in italics. References to papers expressly treating of the subject named are likewise in italics; and casual notices, giving but little information, are usually indicated by bracketed page-numbers. The titles of papers presented, but not printed in this volume, are followed by bracketed page-numbers only.]

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77	Bottom.	Footnote [4]. For "p. 478," read "p. 400."
80	1	For "3 tons," read "5 tons."
82	6	For "small," read "large."
89	27–32	Omit entire paragraph.
93	18	After word "ore," omit remainder of line, and lines 19 to 21.
303	1	For "Cobalt," read "Toronto."
809 } 811 }		Footnotes [5] and [9] should read, "*Applied Science* (Transactions of the University of Toronto Engineering Society), New Series, vol. i., No. 3, p. 125 (Jan., 1908)."